# NOTES ON METALLURGICAL MILL CONSTRUCTION

EDITED BY

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## PREFACE

This book is a reprint of a series of articles, bearing upon some of the important details that enter into the construction of metallurgical plants, especially mills of various kinds, which have appeared in the *Engineering and Mining Journal,* chiefly during the last three years; in a few cases articles from earlier issues have been inserted, in view of their special importance; and there is one article from the *Pacific Coast Miner.* Some of the articles are abstracts of papers originally presented before engineering societies, subsequently published in the *Engineering and Mining Journal,* as to which proper acknowledgment has been made.

The articles comprised in this book relate to a variety of subjects, which are of great importance in the design, construction, and operation of metallurgical mills, but have not been treated with any fullness in technical literature, save in the periodical literature and transactions of the engineering societies, wherein they escape general availability. For this reason, it has appeared useful to collect and republish in convenient form the articles of this character which have previously been printed in the *Engineering and Mining Journal.*

I recognize fully the deficiencies of the present work, not referring, however, to the value of the articles herein assembled, but rather to their heterogeneous character, and the absence of contributions on many important subjects, which would be necessary to round out a general treatise on metallurgical mill construction. I hope, therefore, that the readers of the book will appreciate my intention, and will accept the book simply as a series of notes and essays, covering some of the principal subjects upon which the engineers of four continents have written during the last three years.

W. R. Ingalls.

June 1, 1906.

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# PART I

# BRICKWORK AND CONCRETE

## COST OF EARTH WORK

(September 19, 1903)

H. P. Gillette, in his recently published treatise on "Earth Work and Its Cost," summarizes various data as to the cost of handling earth by picking and shoveling in the statement that the cost of excavating with pick and loading is about 40c. per cu. yd. for hardpan; 20c. per cu. yd. for tough clay; 15c. per cu. yd. for ordinary clay, gravel or loam, and 12c. per cu. yd. for very light sandy or loamy soils, wages being reckoned at 15c. per hour in all cases. After earth is once loosened and shoveled upon a board platform, as in casting in stages out of a deep trench, one man will shovel off the boards all that two men can loosen and cast up. Although a man can possibly cast earth about 12 ft. vertically from floor to floor, it is best to have floors only 5 to 7 ft. apart. The quantity of earth that a man can handle with a shovel varies not only with the character of the soil, but also with the method of attack. If a man is shoveling from a face of earth over a foot high, one that he can readily undermine with a pick, he can load 1.8 cu. yd. per hour on an average, but if he is shoveling plowed soil, where he must use more time to force the shovel into the soil, his output will be only about 1.4 cu. yd. per hour. If he is shoveling loose earth off boards upon which it has been dumped his output is about 2.5 cu. yd. per hour.

The size of the shovel makes a marked difference in the efficiency of the laborer. A small, round-pointed shovel must be used in tough soils, but nothing other than large, square-pointed scoops should be used in handling earth off boards or in shoveling sand, unless it is to be cast some distance. With a large, square-pointed scoop a strong man can load sand into a wheelbarrow at the rate of 1 cu. yd. in five minutes. Roughly speaking, it takes 150 to 250 shovels of earth to make 1 cu. yd.; in casting into a wagon-box at a good steady gait seven shovels are loaded per minute. This is for a vertical lift of about 5 ft., but in casting

out of a trench with a vertical lift of 10 ft., only five shovels are cast per minute. In casting earth horizontally, nine shovels per minute may be done for a 5-ft. throw and about one-half as many for an 18- or 20-ft. throw. With wages at 15c. per hour, it costs about 5c. to carry a cubic yard 10 ft. in shovels, hence men should be close enough to the wagon they are loading, so as not to have to take any steps before casting. The further away a man is from the wagon the fewer shovels can be cast in a given time, and as each shovelful is also smaller, a man 12 or 15 ft. away from the wagon will load only about one-half as much as if he were within 4 or 5 ft. of it; hence it does not pay to crowd more than six men around a wagon, acting upon the idea that quick loading saves money by saving team time. Large square-pointed shovels should be used wherever possible. The work should be directed to a face wherever possible, and a temporary floor should be laid down at the face, so that earth picked down will fall on the floor, whence it can be easily shoveled. A high face of earth should be undermined with pick and then wedged off by driving in bars from the top. Wherever possible, in earth work, men should be paid by the cubic yard, not by the day.

## BRICK MASONRY

By W. R. Ingalls

(August 22, 1903)

Brick masonry is frequently paid for by the cubic foot, and in making estimates it is always reckoned thus, though the result may be converted into the number of thousand brick required, since brick are purchased by the thousand. The cost of brick masonry depends upon the prices for the material and wages of labor, but also to a large extent upon its character; that is, whether laid with thick mortar joints or thin ones, with lime mortar, lime and cement mortar, or cement mortar, and the character of the workmanship in laying the brick. In massive masonry laid with mortar joints $\frac{1}{4}$ to $\frac{3}{8}$ in. thick, the mortar constitutes about 20 per cent. of the entire mass. The effect of thicker or thinner joints upon the cost will be as the relative price per cubic yard of brick and of the kind of mortar used. The size of the brick is very important in considering the cost of masonry. The brick used in some parts of the United States measure only 60 cu. in., while those used in other parts measure 80 cu. in. Of course, a cubic foot of masonry will comprise fewer of the large brick than of the small, while the labor cost of laying a thousand will be the same in each case. Moreover, large brick sell at the kilns where they are made at prices no higher than the small brick made elsewhere. This is because the raw material is of comparatively little value and the labor in manufacture is largely determined by the number of pieces handled rather than their weight. Common brick are rarely used otherwise than locally in the districts which have their own sizes, which is probably the reason why we have no national standard. With fire-brick the case is different. However, it would appear that there might be some advantage in having some uniformity in red brick, such as a set of standard sizes, one of which might correspond with the standard fire-brick, having a volume of about 100 cu. in. Even a large

brick might be desirable for some purposes. One of the advantages of the Custodis method of chimney construction is the reduced cost of laying the large, specially shaped brick used therein. One of the reasons for the use of small brick as compared with large ones, say 16 in. by 8 in. by 4 in., like the Mexican *adobe,* is the better bond, but in that respect I do not suppose there would be material difference between brick of 60 cu. in. and those of 100 cu. in. Another reason for the use of small brick is the thinner walls they permit, which is a matter of some consequence in the construction of buildings in cities where land is highly valuable.

## SAND-LIME BRICK

(October 27, 1904)

According to a paper prepared by S. V. Peppel for the United States Geological Survey, there are in this country at present about 50 plants, with a total capacity of approximately 1,000,000 bricks a day. The experience of these plants indicates that sand-lime brick can usually be manufactured at a cost below that of common clay brick. Sand-lime bricks have been in use long enough, both in this country and in foreign countries, to prove that when properly made they have sufficient strength and sufficient water and weather resisting qualities to make them a safe building material.

The sand-lime brick is the natural outcome of improvements made in the old mortar brick, which has been known for years. This mortar brick was at first never more than a molded mixture of lime and sand mortar, which was allowed to harden in the air. About twenty-five or thirty years ago, one Dr. Michaelis patented a process for the hardening of mixtures of lime and sand by steam under pressure, which is the fundamental principle on which the manufacture of sand-lime brick is based.

The commercial development of the industry dates back only fifteen years in foreign countries, and not more than four years in the United States. In 1896 Germany had only five factories where sand-lime brick was made, but now it has about 200, with an actual annual output of between 350,000,000 and 400,000,000. Early in 1901 a plant was built in Michigan City, Ind. In 1902 about 20 plants were in existence and 6,000,000 bricks were actually sold. Full data are not obtainable as to the actual output in 1903, but about 20,000,000 bricks have been reported as sold in that year. Many of the factories had just started, and were not manufacturing to their full capacity during the year.

## CONCRETE MIXTURE

(November 18, 1905)

In proportioning a mixture for concrete or mortar, the engineer of experience uses those proportions for different work which he has found most effective. For important work, however, where local materials are used, it is always well to determine the proportions by actual test. This is not troublesome, nor is it as expensive as might be supposed. The instruments required for the test are an old oil barrel and a dormant scale. The barrel is shoveled full of sand and "struck" to give a flush top surface. It is then weighed; and, after weighing, sufficient water is turned into it to bring the level of the sand and water to the top of the barrel. It is then weighed again, the difference in weight giving the amount of water required to fill the voids in the sand. A similar procedure, with crushed stone (or gravel) and water, gives the weight in pounds of the water required to fill the voids in the aggregate. As the cement and the cement and sand matrix have not the same specific gravity as water, for each case it is necessary to reduce the weight of the water to units of volume, the result giving both the volume of the cement alone and also that of the matrix required to fill the voids completely. The sand and the aggregate, when placed in the barrel, should be, as nearly as possible, in the same condition of density as they are to be when mixed.

## REQUIREMENTS FOR CONCRETE

**(March 31, 1904)**

S. B. Newberry, in a paper read before the Indiana Engineering Society at Indianapolis, Ind., Jan. 14, 1904, stated that nowadays most engineers use crushed stone just as it comes from the breaker. without screening. However, good quartz gravel is better than stone, since it is harder (save in the case of quartzite or trap rock) and its round form leaves less voids. Round stone for cement does not make a weaker concrete than that made with angular material.

The proportion of voids is of great importance, and insufficient attention is given to it. The strength of concrete is proportional to the ratio of the percentage of cement to voids. Mixtures in which the voids are filled to the same extent will be approximately equal in strength. Superficial area is also an important factor, since all surfaces must be coated with a film of cement in order to produce proper adhesion; the greater the surface the greater the quantity of cement required. Coarser materials give a greater strength than fine materials, chiefly for this reason. Machine mixing is to be recommended; the sand and cement should first be mixed, the water should then be added, and finally the stone or gravel, which should be previously well wetted.

Portland cement concrete suffers no harm by freezing after the mass has fully set; its hardening is interrupted, but proceeds again without hindrance after thawing. Damage is to be feared from frost before setting, however, especially if an excess of water be used.

## LIMESTONE SCREENINGS FOR USE IN CONCRETE

(September 1, 1904)

The advisability of using stone-crusher dust as a substitute for sand in the aggregate for concrete is the subject of a good deal of discussion among constructing engineers. It is of considerable interest to mining and metallurgical engineers both with regard to structural work and the possible sale of mill tailings in favorable localities.

G. J. Griesenauer has stated (*Engineering News*, July 28, 1904) that tests in the laboratory of the Chicago, Milwaukee & St. Paul railway have shown that limestone screenings are far superior to sand as a concrete aggregate. Screenings that will at least all pass a 4-mesh sieve but not finer than an 8-mesh are the most valuable. A finer mesh than eight tends to give a product containing an excess of impalpable stone dust, especially if a soft limestone be employed, while a coarser mesh than four gives a product of which the handling is likely to separate the coarse and fine particles to some extent.

Crusher-run limestone would be better than screened crushed limestone if it were practicable to avoid a separation of the coarse and fine, but this being impossible the product is not uniform and therefore is likely to produce a non-homogeneous concrete. Consequently it is preferable to screen the crushed stone and keep the coarse and screenings separate, mixing them in the proper proportions when required.

The quality of the limestone affects the value of the screenings, very soft stone being naturally inferior to the harder kinds. A very soft limestone should not be reduced to as fine screenings as a harder stone. There is nothing to indicate that limestone screenings may weaken after a certain age any more than sand, the particles in each case being enclosed in cement, whose tendency is to increase in strength with time.

## RUBBLE CONCRETE

(August 22, 1903)

The merits of rubble concrete were discussed in the *Engineering News* of July 16, 1903. This is a type of masonry that has developed within the last few years, concerning which little has yet been printed. The term covers two classes of masonry, one being concrete in which large stones are imbedded, and the other being rubble masonry, in which concrete replaces ordinary mortar. Concrete in which stones are imbedded is usually mixed very wet and deposited in a layer into which large spalls, or even one-man stones, are rammed in. Where the spalls are flat bedded, a dry mixed concrete is sometimes used, the spalls being laid on their flat faces and concrete rammed into the vertical joints. From the last mentioned method it is but a step to the kind of masonry in which large irregular stones are deposited in the walls by means of a derrick, the joints between them being afterward filled with concrete, preferably wet mixture. The advantage of this kind of masonry is, of course, its economy, a portion of the concrete being replaced by stone, which is cheap, the saving varying according to the proportion of cement used in the concrete.

One of the greatest advantages of ordinary concrete masonry over ordinary stone masonry is the saving of labor, that of stone cutters being eliminated and a less skilled class of labor being required for laying it, which advantages are partially offset by the necessity for crushing the stone, the necessity usually of providing frames or molds, and the requirement of more cement per cubic yard than in other classes of masonry. The saving is most in comparison with cut-stone masonry. In making comparison between concrete and rubble masory, the conditions are different.

Second-class retaining walls on the Erie Canal required about 0.6 barrel of portland cement per cubic yard of masonry, the mortar being 1:2, whereas a 1:2:5 concrete (packed measure) required 1.1 barrel of cement per cu. yd. With cement at $1.60 per barrel, this makes a difference of about 80c. per cu.

yd. in favor of the rubble, in addition to which there is a saving of about 50c. per cu. yd. in the item of forms and 30c. a cu. yd. in the item of stone crushing, making a total of $1.60 in favor of the rubble. The cost of laying the rubble was about 80c. per cu. yd., using high-priced masons, as against 60c. for mixing and laying concrete by hand, or 40c. by mechanical mixers. These costs of laying indicate clearly that ordinary rubble masonry costs very little more to lay than does concrete. If rubble concrete be used no skilled masons need be employed, so that this slight advantage of concrete over rubble disappears, *Engineering News* argues, therefore, that in localities where stone exists concrete has ordinarily no economic advantage over stone masonry, except in places where cut stone would be used, and is at a positive disadvantage as a backing where appearance is of no consequence.

With respect to the two varieties of rubble concrete, for small retaining walls and for comparatively thin foundations, the method of ramming large spalls into wet mixed concrete is doubtless to be chosen. In massive masonry where the quarries yield large blocks of stone, the blocks should be bedded in soft concrete and the vertical joints filled with soft concrete into which spalls may be rammed. Where stone comes from the quarry with natural flat beds, common mortar may well continue to be used in the bed-joints, the vertical joints being filled with concrete, which will avoid dressing the stone at all. Where stone comes in slabs easily broken by a hammer, or where retaining walls are too thin to permit the use of large blocks, a soft concrete into which the small stones are rammed may be used. Where stones come out in large, tough, irregular blocks a true rubble concrete is probably the cheapest, since even with the employment of skilled masons the cost of laying is but slightly greater than mixing and laying concrete with common labor.

As between ordinary concrete and rubble concrete, if for any reason great strength is required, the latter still has the advantage, since if the cement saved by imbedding large stones in the concrete be used to make a richer mortar, the rubble concrete will be superior in strength to the ordinary concrete, the latter having the weaker mortar. The writer concludes his argument with the statement that a dollar will buy either more rubble concrete than ordinary concrete, or will buy a stronger masonry.

## CONCRETE WORK ABOUT MINES[1]

BY HENRY W. EDWARDS

(May 19, 1904)

Concrete, with well-proportioned ingredients, may be safely relied upon for a crushing strength of 50 tons per square foot. The ingredients are crushed stone, sand, cement and water. The stone is usually crushed to pass a 2 to 2.5-in. ring; soft stone, such as decomposed porphyry, is objectionable, unless the concrete is to bear a light load. Slag broken to the same size is in every way suitable; it has only one objectionable ingredient, namely, calcium sulphide. The broken stone or slag is not improved by screening. Jig tailings, both coarse and fine, give good results when used for sand, a mixture of all sizes, with 0.05 to 0.09 in. predominating, being best. Vanner tailings and stamp-battery tailings are usually too fine. Granulated slag is excellent. As regards cement, the so-called natural cements, while cheaper in first cost, are usually less effective than portland cements. Any doubt as to whether a particular sample of cement is natural or artificial can usually be settled by testing for magnesia. In portland cement this is present as an accidental impurity, about 2 per cent., while in natural cement it sometimes exceeds 10 per cent. Generally speaking, concrete containing two parts of portland cement equals three parts of natural cement. Cement should be finely ground, but not to exceed 100-mesh. In testing through wire-cloth of this fineness, it is important to see that the meshes of the sieve have not been distorted, allowing coarser material to pass. There is a marked difference between various brands of portland cement, and these differences are accentuated by age and care in storage.

Tests of the tensile strength of cement are easily made by an ordinary 5-ton platform scale, with a good, strong screw-jack;

[1] Abstract of a paper on "Concrete in Mining and Metallurgical Engineering" read before the American Institute of Mining Engineers, February, 1904.

the pressure exerted by the screw-jack is read on the beam of the scale at the moment cracks appear on the visible sides of the test piece. A series of mixtures should be made with each brand of cement, these mixtures being rammed in a wooden box 9 in. square by 30 in. long, and left there for several days, or weeks, until set. Many tests should be made of each series of mixtures, these mixtures ranging from one part of cement with 0.5 part of sand or tailings and seven parts unscreened crushed rock, to one part of cement with four parts of sand and three parts of rock.

Concrete may be divided into three classes, according to the work required of it: (1) Strong, containing about 15 per cent. of cement, for retaining walls, flues, culverts, arch work in general, and foundations in wet places; (2) medium, containing about 10 per cent. of cement, for engine, machinery, stack, and furnace foundations, on good, dry ground, and for the bottoms of flues; (3) poor, containing about 7 or 8 per cent. of cement for leveling the bottoms of excavations previous to beginning foundations proper, and for all foundations and below-ground work, where the weight to be supported will not exceed 6 tons per square foot. Both medium and poor grades may be diluted further without diminishing the ultimate strength, by using large boulders, provided the concrete is properly tamped.

In testing, it is only necessary to work on some of these varieties, usually the strong. One or two blocks of each series may be tested at the end of a week, leaving the others to be tested at the end of four weeks. Any block which stands a 2-ton breaking test can be pronounced very good, although a 4-ton test is not too much to expect after several months' drying.

It is to be noted that two parts of sand, four parts of crushed rock, and one part of cement mixed together do not give six volumes of mixture, as the sand fills the spaces between pieces of rock, and the cement those between the sand and rock. On thoroughly ramming this concrete in place it will pack to a bulk of four volumes, or, approximately, the original bulk of the crushed rock.

Generally speaking, machine mixing is better than hand mixing, and it will pay to install a mixer if more than 80 cu. yd. is required. An efficient mixer is a cubic wooden box, lined with No. 10 sheet iron, and having an iron manhole at one corner. The box is mounted on two trunnions, one a piece of 3-in. pipe

through which water is introduced, the other connected by a gear-wheel and pinion to a hand crank. The box is given a few revolutions to mix the ingredients dry, the necessary quantity of water is introduced by a hose and nozzle through the hollow trunnion, and then the box is revolved as long as desired. Ideal mixing is to have each particle of rock and sand coated entirely with cement. In hand mixing by shovels, a sheet-iron platform lightens labor.

The necessary amount of water depends upon the climate, the character of the other ingredients, and the intended use. A good rule is to have the concrete so wet that it will shake like jelly when being rammed by a heavy beater, but in shallow layers, as in floors, the mixture should be much wetter than when laid in mass, otherwise it will dry up before chemical action begins. In retaining walls, it is well to use the mixture reasonably dry, since an excess of water, on evaporation, leaves the concrete porous. For tanks, reservoirs, etc., the mixture should contain more concrete, and should be more thoroughly mixed and tamped.

Concrete should be mixed as near the point of use as possible, and a long trip in a wheelbarrow is to be avoided. Where it must be handled by wheelbarrow, it is best to dump the material on a small platform and shovel it over, before putting it in place. Care should be taken that all parts are equally wet; otherwise some portions of the work may dry before others, and thus cause cracks.

For foundations and walls, the material should be laid in layers, not over 6 in. deep, to get the full benefit of beating. A convenient beater is made of an iron casting 6 in. square, with a handle of 1 or 1.5-in pipe about 5 ft. long, the whole weighing 20 to 25 lb. Each layer should be put on before the previous one is set, and not more than a half-hour should elapse between mixing and depositing. It is advantageous to have the work continue day and night, but, if interrupted, the layer that has set should be cleaned and wetted first with water, and then with thin cement grout, before adding a new layer. As vertical joints in a wall are less weakening than horizontal ones, a long retaining wall may be divided into panels, each of a size that can be completed in a day's work.

The making of cribs or forms permits much ingenuity. They should be stiff enough to stand the beating of the concrete, and

so arranged that the timber may be used repeatedly. Plain 2-in. planks, with straight planed edges, are better than tongue-and-groove boards. In filling behind such a crib the concrete should never be dumped from a height, else the mortar and stone will separate. In a deep excavation the concrete may be lowered in a tub or bucket and dumped in place.

The time required for seasoning varies with the climate and the season of the year. In summer, or in a dry winter climate, the action is rapid, and a covering of moist sand may be necessary to prevent too rapid hardening. In northern climates with cold winters, concrete laid in the fall will not be really solid until warm weather sets in. Where the load is to be applied gradually, as in a retaining wall filled from behind, loading may begin after a week or two of warm weather. For thin work, such as floors and flues, a few days may suffice; an engine foundation can be used after a week or two. In heavy masses, hardening and seasoning may go on for many months.

For work under water, a home-made iron funnel, with a stem long enough to reach bottom and cut off as necessary, is useful. By resting the outlet of the funnel on the bottom, filling the stem and hopper with concrete, lifting a little and moving as desired, the concrete can be spread evenly. In running water, a coffer-dam, slightly higher than the depth of water, may be made out of a double layer of boards, with a layer of tarred roofing paper between. The ground upon which the foundation is to rest is leveled roughly, and the coffer-dam is carefully sunk into position by loading it with concrete. This is much cheaper than pumping out the water and laying the concrete in the usual manner; it is cheaper also than sheet piling.

It is difficult to give any definite figures of cost, owing to the great variation in the price of the ingredients in different localities; also, the size of the work is a factor. Presuming, however, that the materials are delivered in railroad cars convenient to the work, I find that an average cost covering unloading, mixing, arranging platform, placing and beating is approximately equivalent to a cubic yard of structure per man per day. This covers incidentals, such as wear and tear on tools, etc., but not the cost of cribs, excavations, scaffolding, lumber, etc. The cheapest piece of work of which I have record is a retaining wall 16 ft. high, 92 ft. long, and 3 ft. thick at the base, tapering to 20 in.

at the top. Jig tailings and picking-belt rock available on the spot were used, while cement cost $2.15 per bbl., lumber $14 per 1000 ft., and wages were 15c. an hour. The total cost was 22c. per cu. ft., including supervision. A large engine foundation of hand-broken slag, granulated slag and cement cost, including crib, $7.75 per cu. yd. of finished work.

## CONCRETE FOUNDATIONS AND FLOORS[1]

In construction, the men executing masonry or concrete work will often go to considerable trouble to secure screened sand, irrespective of the really positive advantage that is to be obtained in most cases by using unscreened material. Sand that will pass an 8-mesh sieve ought never to be screened except for the highest class brick or cut-stone (coursed) masonry.

Well made 1:5 portland cement concrete, of good standard gravel or limestone screening, will have a compressive strength at four weeks of over 2000 lb. per sq. in.; and at one year, over 3000 lb. Tests made at the Case School of Science showed that 3-in. cubes of ½: 6 portland cement and gravel concrete had a compressive strength of 3200 lb. per sq. in. at six weeks, a sp. gr. of 2.17, and an absorption of water of 4.16 per cent. Similar cubes of 1½ parts cement, ½ hydrated lime, and 6 of sand and gravel had a strength of 3880 lb., a sp. gr. of 2.18, and an absorption of 4.10. Cubes of 1½ parts of cement and 6 of limestone screening, poured in a porous mold, had a strength of 2000 lb., a sp. gr. of 2.5, and an absorption of 5.04.

The coefficient of expansion of concrete, of the proportions 1:2:4, by heat, has been determined as 0.0000055 for 1 deg. F., which is almost the same as that of untempered steel, which is 0.0000060.

In proportioning the material, when the mixture is by volume and the mixing is done by hand, a bottomless box will serve as a simple and effective measure. If the box is made of 2×12-in. plank, 3 ft. square inside, the dimensions are most satisfactory. Such a box, once even-full of sand, and evenly filled three times with the aggregate, with four bags of cement, will make just 1 cu. yd. of concrete (of the proportion 1—2—6), in place. The cement, of course, is to be mixed with the sand before adding the aggregate. Four short pieces of wood are spiked fast to the box, serving as handles.

[1] An arrangement of miscellaneous notes from the *Engineering and Mining Journal*, Vol. LXXX.

Concrete, made of the "run-of-crusher" rock, without the addition of sand has been successfully used; when the particles, known as "dust," are sufficiently clean and sharp, the results are even better than when sand is used.

Concrete made of broken or crushed brick instead of stone gives excellent results for some purposes; but it always requires a larger proportion of cement than when less porous material is used.

The writer once saw a gang of men mixing concrete by hand, with pointed shovels. This statement may not bring to mind the actual heinousness of the crime against good practice that such work becomes, but when one considers that square-toed shovels would pick up and mix with the aggregate the wet cement and sand (the most costly ingredient of the batch) that the pointed or round-toed shovel allows to escape, the term does not seem too strong. The square-toed D-handled shovel is about the only one that is suited for such work as this, and should always be used in preference to any other form. When mixing is done by hand, the mixing board should always be used, unless a smooth, impervious floor may be at hand. It seems needless to caution against mixing on the bare ground, and yet it is done more often than one would expect, through ignorance or carelessness. The amount of cement that is lost in this practice is beyond calculation, and criminal from the standpoint of economics.

For floors a mixture of 1 part portland cement, 3 parts clean, sharp sand and 5 parts of broken stone that will pass through a 2-inch ring is recommended. There should be a finishing coat ½ to 1 in. thick, according to amount of use of the floor, and this may be made of various proportions, from 1 of cement and ½ of sand, to 1 of cement and 2 of sand, the mixtures with the larger amount of sand standing the wear of traffic best, and those richest in cement being most nearly impervious to water. Trowel finish usually makes a floor quite impervious. Expansion joints must be provided if there is likely to be considerable change in temperature of the floor, or steel mesh reënforcing may be used. Such a floor is not ready to use in less than a week after it is fully completed, and longer time is desirable. It will set sufficiently to bear weights in less time, but will not stand wear. Failure may occur even with the utmost care and the best ma-

terials. It is always advisable to keep the upper surface damp while the material is setting.

Concrete floors are not suitable for use about electrolytic or lixiviating plants because acid solutions attack and gradually disintegrate the cement; but this action can be largely prevented by asphalt coatings.

In placing concrete in two or more layers, which is the usual practice in floor or sidewalk work, it is desirable to place the top layer before the first is set, or, in case this rule cannot be followed, a firm bond is secured by thoroughly scratching the first layer while it is soft.

Care should be taken during hot weather to prevent the rapid drying of concrete. It should be protected from the sun and occasionally sprinkled with water; it is a good plan to cover the smaller masses with a wet cloth. This will prevent, to a large extent, the formation of "hair cracks," or crazing.

# PART II

# BUILDING CONSTRUCTION

## DESIGN OF MILL BUILDINGS

(March 31, 1904)

The best modern practice inclines toward single-floor buildings for shop and factory purposes; the advantages of this type over multiple story buildings are better light, better ventilation, easier heating, cheaper foundations, absence of vibration, cheaper floors, better supervision, cheaper handling of material, cheaper construction, less danger of damage by fire, and better ability to make extensions. The floors may be laid with cinder, or with brick on a gravel or concrete foundation, but in buildings where men have to work at machines the favorite floor is plank on a foundation of cinder, gravel, or tar concrete. Concrete or cement floors are used in many cases with good results, but are unsatisfactory where men have to stand at benches or machines.

Buildings in which men are to work should be well lighted. It is now the common practice to make as much of the roof and sides of a transparent or translucent material as practicable; in many cases 50 per cent. of the roof surface is made of glass, while skylights equal to 25 or 30 per cent. of the roof surface are very common. Windows and skylights directly exposed to the sunlight should be curtained with white muslin cloth, which admits much of the light and prevents the glare and excessive heat in summer, which are objectionable. The saw-tooth type of roof with the shorter and glazed tooth facing the north gives the best light, and is now coming into general use. The principal difficulty in saw-tooth roof construction is the arrangement of satisfactory gutters and the liability of snow to drift on the roof.

Plain glass, wire-glass and ribbed glass are used for glazing the windows and skylights of factory buildings. Ribbed glass should be placed with the ribs vertical, since otherwise the glass gives a glare which is very trying. Wire netting should always be stretched under skylights of ordinary glass, to prevent accidents from falling glass in case of breakage.

## COST OF A SINGLE-FLOOR MILL

(April 26, 1902)

A workshop, 120 by 180 ft., with a single floor at ground level, recently erected in Massachusetts, cost 65c. per square foot. The framing conformed to the principles of standard mill construction. The roof was three-inch plank, tarred and graveled. The sides were closed in with windows of translucent glass, hinged from the top and swinging outward, each window filling a bay down to about three feet from the ground. Below the windows the sides were concrete, laid on expanded metal. Additional light was provided by monitors in the roof. The floor consisted of four inches of concrete, surfaced smoothly, laid on eight inches of clinker, well packed down.

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In making templets for bearings under steel beams, bluestone curbing is cheaper by far than the cut stone commonly used, and is sufficiently smooth and even, in most cases, to answer the purpose. For beams of 12-in. depth and over, however, it is better to use thicker material than the (nominal) 4-in. curbing.

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In the construction of brickwork, the most intricate problems can be solved by the simple expedient of building a model. This is not as difficult or expensive as one might suppose. Soap, cut into blocks $\frac{1}{4}$ by $\frac{1}{2}$ by 1 in. is the most satisfactory material for such models; the ordinary yellow laundry variety gives good results and is the least expensive to use. Where much of this work is to be done, the best way to cut the blocks is by a case-knife, guided by saw-cuts in the cutting board, somewhat in the manner of a carpenter's miter-box

## LIGHTING OF WORKSHOPS

(September 5, 1903)

Good light in a factory or workshop is an important consideration in connection with the obtaining of the maximum efficiency of the labor, but we know of no rule as to the proper proportioning of the windows to floor space, etc. According to C. Zimmermann, M.D., in the *Journal of the American Medical Association*, Jan. 19, 1901, a school-room, to be thoroughly well lighted, should have 30 sq. in. of glass per square foot of floor, or 1 sq. ft. of glass to 5 sq. ft. of floor; the window sill should be at least 40 in. above the floor in order to prevent glare from below, and the walls of the building should be chamfered at the windows, on the inside. These data may be a guide in factory design, but we conceive there are many other factors to be considered, such as hight of the stories, relation between floor space and length or width of the room, exterior obstructions to light, etc. A one-story building 200 by 200 ft. might have windows 10 ft. high all the way round the outside, giving a ratio of glass to floor of approximately 1 to 5, and yet not be thoroughly well lighted. Such a building would doubtless be designed with illumination from above, either by means of monitor sky-lights, or by a roof construction of the saw-tooth type. The character of the glass used in the windows is also a factor. We find in our notebook the statement, authority unknown, that light is diminished 13 per cent. by the interception of polished plate glass $\frac{1}{4}$ in. thick, 30 per cent. by rough cast plate $\frac{1}{4}$ in. thick, 53 per cent. by rough. rolled glass $\frac{1}{4}$ in. thick, and 22 per cent. by 32-oz. sheet glass.

## LIGHTING OF MILL BUILDINGS [1]

By C. A. Raymond

(August 5, 1905)

It is not until recently that the lighting of mill buildings has occupied the attention of designers in this country to any great extent, but rapid strides are now being made in the direction of better diffusion. The area of glass in most buildings has been largely increased, in some instances to excess. The area of glass should be limited by the following considerations: (1) In mill buildings where double glazing is too expensive the heating is rendered more difficult by an increase of glass surface. (2) An excess of glass, when exposed to direct sunlight, is apt to make the building excessively warm, unless rendered translucent by a coating of white lead or similar preparation. (3) The expense of replacing broken glass in some classes of shops may be considerable, though this may be obviated by proper protection.

The location of a building, with respect to the points of the compass, largely affects the quality of the lighting. North light is the best, because of its steadiness. It is, however, not so important to have a large quantity of light as to have it well diffused. These facts have developed the saw-tooth roof, a European type, which is now being used in some very good buildings in this country. There are several strong objections to it, but much also to be said in its favor. It is not difficult to design a building so that this roof may be used no matter how the land may lie with reference to the points of the compass, the worst case being when the designer is compelled to run his building toward the quarter points. In locations where much snow falls, this type of roof should be avoided; but where snows are infrequent and not usually deep, it offers, as a rule, a very satisfactory method of lighting, though some means must often be provided for warming the gutters to dispose of the snow and ice rapidly. Gutters should be specially large or leakage is apt to occur.

[1] Abstract of a paper read before the Toledo Society of Engineers.

As far as possible, roof lights, to be most efficient, should be placed normal to the direction of the source of light. This is illustrated by the usual form of saw-tooth, which is inclined at an angle of 20 deg. to the vertical.

Ribbed glass has done much to improve the quality of light in mill buildings. It serves as a cheap substitute for prismatic glass, and, owing to its great diffusive qualities and low cost, is extensively used in both skylights and windows. It is no doubt most effective for windows when the ribs run horizontally, but shopmen who have worked by it without shades find the glare from it troublesome to the eyes when exposed to direct sunlight. This difficulty may be effectively obviated by shades, but these are ordinarily considered too expensive. With the use of this glass the central part of the room gains a large part of that light which in the case of ordinary glass falls on the floor near the windows.

Translucent fabric is a fairly low-priced and otherwise good substitute for glass. Being translucent and not transparent, it is good for skylights.

In order to aid in the diffusion of light, the interior of the building may be painted white, simple whitewash being frequently employed.

Condensation is a point to be reckoned with in skylights, but is usually cared for by means of gutters placed at intervals underneath the glass. Many kinds of roofing, also, are troublesome on this account.

## HOLLOW BRICK FOR MILL-BUILDING CONSTRUCTION

(March 31, 1904)

Mill-buildings for machine shops, foundries, metallurgical works and similar purposes are commonly constructed of steel framework, closed in with corrugated iron; or of brick walls and steel roof. The latter construction is the more expensive, but it is also the more substantial, and possesses other advantages. A combination of the two kinds of construction has also come into extensive use, viz.: a self-supporting steel frame with curtain walls of brick or concrete laid in between the steel columns; these walls can be made much thinner than would be otherwise required. A more recent improvement is the construction of such curtain walls of hollow brick or tile, which effects a large saving in material and labor, and has been proved to be an entirely practicable system of construction, an example of which is to be seen in the tank-house of the American Smelting and Refining Company's works at Perth Amboy, N. J.

This is a building 350 ft. long, 200 ft. wide, and 24 ft. 6 in. high. The roof is supported by columns (12-inch I-beams) spaced 16 ft. apart, centers. The spaces between the columns were laid up with hollow tile, 12 by 8 by 4 in., set so as to make a wall 4 in. thick. Between every second course of blocks a strip of band iron, 1 in. by $\frac{1}{8}$ in., was laid in the cement joint and secured to the I-beams by 1 in. by $\frac{5}{16}$-in. rivets. A. Muller, in *Insurance Engineering*, April, 1902, compares the cost of this construction with that of a solid brick wall, 1.5 brick thick, as follows:

Solid brick: 565 M of brick (21 per sq. ft. of wall) @ $5.75 per M, $3250. Cement, sand, and labor of laying, @ $8.57 per M, $4841; total, $8091.

Hollow brick: Contract price delivered, including band iron, was $1800; cement, sand, and labor, @ 6c. per sq. ft., $46.20 per M, came to $16.17; total, $3426.

The difference in favor of the hollow blocks was, therefore,

$4665, but this does not take into account the cost of the steel columns and their erection. The weight of the 565 M of common brick was 1130 tons, while that of the 35 M of hollow blocks was only 280 tons.

All of the buildings at the new plant of the Barber Asphalt Paving Company in New Jersey have been constructed on a similar system, in this case called the Phœnix system, which has been patented by Henry Maurer & Son, of New York. The hollow blocks, 12 by 8 by 4 in., are made of hard-burned terra-cotta. The wall panels are the spaces between I-beam columns set 15 ft. apart, centers. The wall is reinforced by a strip of band iron laid in cement between each course of blocks. The comparative cost of a wall of common red brick and one of the Phœnix hollow tile, at prevailing prices in the neighborhood of New York City, is estimated in *Insurance Engineering*, October, 1903, as follows:

Solid brick: A mason working 8 hours, at 65c. per hour, with a helper at 37.5c., will lay 1200 brick, making the labor cost $6.33 per M. The brick cost $7.25 per M, and 1 bbl. of cement, at $2.50, 1 bbl. of lime, at $1.25, and 1 cu. yd. of sand, at $1.25, are required for the mortar, making the total cost of the brick in the wall $18.58 per M. Reckoning 21 brick per cu. ft., or per sq. ft., if the wall be 12 in. thick, the cost of 100 sq. ft. of wall is $39.02, or 39c. per sq. ft.

Hollow brick: The construction of a 4-in. wall 15 ft. high, with I-beam columns 15 ft. apart, requires for each 225 sq. ft. one 6-in. I-beam, weighing 227.25 lb. (14.75 lb. per foot), 350 ft. of 1-in. by $\frac{1}{16}$-in. band iron, weighing 0.09 lb. per foot, 330 hollow blocks, 12 by 8 by 4 in., 1.5 bbl. cement, and 0.75 cu. yd. of sand. The cost of this material is as follows: 227.25 lb. of I-beam, at 2.65c., $6.02; 31.25 lb. band iron, at 3.2c., $1; 1.5 bbl. cement, at $2.50, $3.75; 0.75 cu. yd. of sand, at $1.25, 94c.; 330 hollow blocks, at 8.18c., $27; total, $38.71; equivalent to about $1.55 per sq. yd. and 17.2c. per sq. ft. for material alone. A mason, working 8 hours at 65c. per hour, with a helper at 37.5c. per hour, will lay 220 blocks, making 150 sq. ft.; wherefore the labor cost is about 5.5c. per sq. ft., and the total cost of the wall is 22.7c. per sq. ft., exclusive of the cost of scaffolds, which will be the same in either case.

A common red brick weighs about 4 lb., while a hollow brick,

12 by 8 by 4 in., weighs 16 lb. Inclusive of the iron and steel, the ratio of weight of a 4-in. hollow brick wall to a 12-in. solid wall is about 1:3.3, which permits a saving to be made in the foundations. The other advantages of hollow walls are adequate strength, thorough fireproof character and quickness of erection. For many kinds of work, such a building, with a roof of red clay tiles, makes an ideal construction. A recent invention is the manufacture of glass tiles, which can be laid in connection with red clay tiles so as to give admirable lighting from above.

## NEW USES OF CONCRETE IN BUILDING CONSTRUCTION

(October 24, 1903)

One of the important factors in connection with the recent remarkable development of the American cement industry is the increasing attention which is being directed to the use of concrete in building construction, chiefly in the form of reinforced concrete. The ease with which concrete can be molded into monolithic forms, its great strength, the absolute protection which it gives to iron and steel imbedded in it, and its fireproof qualities, which have been established beyond doubt, combine to make it a material of the highest value. The idea of reinforced concrete, i.e., concrete strengthened by light steel bars or wire netting, or expanded metal imbedded in it, was first developed in France by J. Monier, and early found application in metallurgical engineering in the construction of dust- and fume-settling flues. These have become standard practice, and are to be seen at nearly all of the most recently constructed lead and copper smelteries in the United States, notably at the new plant of the American Smelting and Refining Company, at Murray, Utah. In general building construction the Monier system has been more generally adopted in France and Germany, but during the last few years its use has become considerable in the United States.

There are a number of patented modifications of the Monier system. Among these may be named the Ransome, in which twisted steel bars are used; the Hennebique, using round steel bars and a certain style of stirrups to resist shearing; the De Valliere, differing from the Hennebique principally in the style of the stirrups employed; the Thatcher, using strengthening bars with projections in connection with the concrete; the Columbian, using special forms of rolled steel; the Roebling, in which a netting and rods are used, principally as centers for concrete arches; the Expanded Metal system, used only for slabs and not for beams; and numerous other less well-known systems. A recently introduced fireproof roofing material is "ferroinclave," made of sand

and cement in the form of slabs 1.5 in. thick, in the center of which is incorporated a sheet of steel, corrugated deeply in a peculiar manner.

The most ambitious example of reinforced concrete construction is the now famous Ingalls building at Cincinnati, which has attracted deep engineering interest. This is a 16-story building, which is practically a monolith of reinforced concrete (Ransome system). It is 50 by 100 ft. in area and 210 ft. in height. The concrete used in its construction is composed of 1 part portland cement, 2 parts sand, and 4 parts of hard limestone of 1 in. size.

Another system of construction that is extending to a marked degree is the use of concrete blocks. An example of this kind of construction is the method of the American Hydraulic Stone Company, of Denver, Colo., described in the *Engineering News* of Sept. 17, 1903. Walls built of concrete blocks are usually hollow, and may be of "one-piece" or "two-piece" construction. In the former case each block is hollow, and its width is equal to the thickness of the wall. In the latter case the blocks are approximately of T-shape in plan, set on edge, so that two blocks form the inner and outer faces of the wall, their ribs, 12 in. apart, serving as cross-bonds. The American Hydraulic Stone Company is exploiting the latter system. The blocks are usually 9 by 24 in. on the face, the ribs varying according to the thickness of wall required. The regular blocks weigh 38 lb. each, and a wall built of them contains 52 per cent. open space. The blocks are molded from a mixture of 1 part cement to 6 or 8 parts of $\frac{3}{4}$-in. crushed stone under hydraulic pressure of 30 to 35 tons. For an ordinary 3-story building, walls of this construction 9 in. thick are sufficient. For a higher building, or for heavy construction, the walls of the lower stories should be 12 or 14 in. thick, or a 10-in. wall may be used, some of the hollow spaces being filled with concrete to form piers. For interior partitions, 4-in. to 8-in. walls are sufficient.

The blocks are made by three styles of hydraulic press — (1) a portable hand press, which will turn out (with five men) 400 blocks per day; (2) a power press, operated by electric motor or gasoline engine (to be used in connection with a power concrete mixer), which will turn out (with six men) 1000 blocks per day; (3) a heavy duty machine, operated by a 10-horse-power engine (serving both for the press and the concrete mixer), which gives 35 tons pressure to each block and turns out (with eight men)

1500 blocks per day. Two masons and a helper can lay 400 blocks in a day.

F. E. Kidder reports that with hand mixing and a hand press, operated by five men, the cost of making blocks $8\frac{3}{4}$ in. by $23\frac{3}{4}$ in. for a square yard of 10-in. wall (12 blocks required), the concrete being 1 part cement to 6 parts sand and gravel, was 0.78 sack of cement (4 sacks to the barrel), 4 cu. ft. of gravel, and the labor of one man for 1.25 hour. Reckoning labor at $2 per 10 hours, gravel at 75c. per load of 1.25 yd., and cement at $2.50 per bbl., the cost per square yard was 84c. Allowing 48c. per square yard for laying and mortar, and 10c. per square yard for carting, the total cost per square yard of wall erected was $1.43 = 15 7-9c. per square foot. As compared with walls of common brick, faced with pressed brick, this is a great saving. It is indeed considerably lower than what the most ordinary brick masonry can be laid for, since a 10-in. wall of this concrete construction will satisfactorily take the place of a 13-in. or 17-in. brick wall. It should be noted also that Mr. Kidder's test was made with hand machines, and naturally the cost of the blocks would be reduced by mechanical mixing and power press. The figures refer, moreover, to so-called "faced" blocks, in the manufacture of which 0.25 in. of the concrete is scraped off the face, which is then surfaced smoothly with 1: 2 cement. When taken from the press the blocks are stacked up for 10 days, and are kept moist by spray from overhead sprinklers.

This appears to be a form of building construction which is well worthy of attention, not only because of its economy, but also because of its advantages in other respects. The hollow walls, divided by partitions, make the building cool in summer and warm in winter, while the spaces can be utilized for heating and ventilating flues, electric wires, plumbing pipes, etc.

# CORRUGATED IRON BUILDINGS

By W. R. Ingalls

(September 19, 1903)

Corrugated sheet iron is one of the most useful and, under many conditions, one of the cheapest of materials for buildings for metallurgical purposes, particularly for roofs. The linear rigidity imparted to light sheets by the process of corrugation makes them self-supporting and gives strength to the light and comparatively inexpensive framing on which they may be used. The framing may be either of timber or steel. The essential parts of such a building are merely the posts, girts, wall plates, trusses, and purlins, on the last of which the corrugated sheets are laid directly.

Corrugated sheet iron is sold either as galvanized or painted. The former is the more expensive, but also the more durable; it is, however, quite unsuitable for buildings intended to contain furnaces which will develop sulphurous fumes, since the latter will quickly corrode the zinc with which the iron is coated, and the galvanic action set up between the remaining zinc and the uncoated iron hastens the destruction of the latter. With any kind of corrugated sheet iron the durability depends upon the thoroughness with which the iron is protected. On this account galvanized sheets are sometimes painted, though commonly used without any other protection than is afforded by the coating of zinc. Ordinary corrugated iron receives one coat of paint at the rolling-mill, the paint usually employed being red oxide of iron, thoroughly ground in pure linseed oil, with enough drier mixed in to give it proper drying qualities. This first coat of paint is applied by machine, and is likely to be imperfect, wherefore the sheets should be painted again after putting them on the building. For this purpose other paints than ferric oxide may be used advantageously, such, for example, as graphite or silica-graphite. Corrugated iron sheets may be obtained from the rolling-mills

painted with graphite or other special paints at an additional cost over the ordinary sheets.

Corrugated sheet iron is sold either by the pound or by the "square" of 100 sq. ft. In calculating the latter the full width and length of the sheets, after being corrugated, is counted; no allowance is made for end or side laps. The approximate weight in pounds per 100 sq. ft. of corrugated sheet iron, with 2½-in. corrugations, of various thickness, per United States standard gage, is given in the following table:

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Gage, No. | 28 | 27 | 26 | 24 | 22 | 20 | 18 | 16 |
| Painted, lb. | 70 | 76 | 82 | 110 | 140 | 166 | 220 | 275 |
| Galvanized, lb. | 87 | 93 | 99 | 127 | 157 | 183 | 237 | 292 |

Standard corrugated iron sheets, with 2½-in. corrugations, are 26 in. wide, and will lay 24 in. wide, with a side lap of one corrugation. They are made in lengths of 6, 7, 8, 9, and 10 ft. When sufficient time (usually about two weeks) is given, sheets may be rolled to special intermediate lengths, but if sheets have to be cut, the next larger length is charged for; thus, an order for sheets 8 ft. 8 in. long would be charged at the price of 9-ft. sheets. The dimensions of sheets of standard sizes and the surface that they will lay is given in the following table:

| | | | | | |
|---|---|---|---|---|---|
| Length, ft. | 6 | 7 | 8 | 9 | 10 |
| Width, in. | 26 | 26 | 26 | 26 | 26 |
| Area, sq. ft. | 13 | 15⅙ | 17⅓ | 19½ | 21⅔ |
| Will lay, sq. ft. | 12 | 14 | 16 | 18 | 20 |

The figures in the last line of the above table make no allowance for end laps, and in estimating must be diminished by the proportion of the latter to the length of the sheets. On siding a 1-in. or 2-in. end lap is sufficient, but on roofing it varies from 3 to 6 in., according to the pitch of the roof.

In laying corrugated iron a nail should be put in at every other corrugation, at the end laps and about every 6 in. on the side laps, nailing through the ridges of the corrugations and not through the furrows. Nails of 1-in., 1¾-in. and 2¼-in. lengths are employed; certain patent barbed roofing nails, which cannot work out, the necks being barbed, may be recommended. In order to make a perfectly tight roof the sheets should have a side lap of one and a half corrugations, whereby water would have to flood up under two full corrugations before it could do

any damage. A side lap of two corrugations is sometimes used, but is no better than a lap of one and a half. In sidings a lap of one corrugation is amply sufficient.

When a corrugated iron roof is to be laid on boards nailed to the rafters, it is advisable to lay waterproof paper between the iron and the boards, especially in buildings where steam or vapor comes in contact with the roof. The paper makes the building warmer and prevents dripping from the roof. Good waterproof paper may be bought at 20c. to 25c. per 100 sq. ft.

If there are valleys in the roof, form a lining from plain sheet iron or steel, painted on both sides, from 18 to 24 in. wide, fit it in the valley, and cut corrugated iron to correspond, lapping the latter from 4 to 6 in. over the valley lining. To cover the comb of the roof, metal ridge caps, usually in lengths of 8 ft., may be obtained. These are made with corrugations to fit into those of the roof sheets, thus making a tight and well-finished roof. Metal corner-boards, casings for window sills, and louver slats are also articles of regular manufacture.

The distance between purlins that is to be spanned by corrugated iron sheets laid directly upon them is determined by the transverse strength of the iron and the load to be sustained. In the parts of the United States where the snowfall is likely to be heavy, roofs are generally made capable of supporting 30 to 50 lb. per square foot, allowing for snow and wind pressure. According to William Kent ("Mechanical Engineer's Pocket-Book," p. 181), it was found by actual trial that No. 20 corrugated iron, spanning 6 ft., began to give a permanent deflection under a load of 30 lb. per square foot, and collapsed under 60 lb. The distance between centers of purlins should not, therefore, exceed 6 ft. for a load of 30 lb. per square foot, and preferably should be less than that. Jones & Laughlin give the following safe loads, in pounds per square foot, for standard corrugated sheets supported by purlins 3, 4, 5, 6, and 7 ft. apart:

| B. W. G. | 3 FT. | 4 FT. | 5 FT. | 6 FT. | 7 FT. |
|---|---|---|---|---|---|
| No. 16 | 135 | 76 | 49 | 34 | 25 |
| No. 18 | 102 | 57 | 37 | 25 | 18 |
| No. 20 | 73 | 41 | 26 | 18 | 14 |
| No. 22 | 58 | 33 | 21 | 14 | 10 |
| No. 24 | 46 | 26 | 16 | 11 | 8 |
| No. 26 | 38 | 21 | 13 | 9 | 6 |

The above figures give a factor of safety of 4. Corrugated iron sheets are the stiffer the larger the corrugations. The transverse strength of corrugated iron is computed by the formula, $W = 99/900\,TBD \div L$, in which W is the breaking weight in pounds, T the thickness of the sheet in inches, B the breadth of the sheet in inches, D the depth of the corrugations in inches, and L the unsupported length in inches.

The cost of an iron and steel building depends chiefly upon the total weight of the material required in its construction. This will correspond approximately to the area of ground covered, and the latter is a convenient basis for rough estimates. Generally speaking, the complete cost of a one-story iron building, such as would be suitable for a furnace shed, storage house, etc., is 40c. to 60c. per square foot of ground covered. A building 309 by 42 ft., weighing 10.5 lb. per square foot, cost 44.65c. per square foot. Other buildings, weighing 15 lb. per square foot, cost 57.5c. and 60c. These costs did not include the footings or pavements of the floors. They figure out to 4c. to 4.25c. per pound, and were done at a time when steel was high (1900 and 1901).

## IRON AND STEEL BUILDINGS

(January 14, 1904)

According to Professor Ketchum, in his treatise on the design of steel mill buildings, the minimum size of angle bars to be employed in the construction of such buildings should be 2 by 2 by 0.25 in., and the minimum thickness of plates 0.25 in. for both permanent and temporary structures. Wherever the metal is to be subjected to corrosive gases, as in train sheds and smelter buildings, the ordinary allowable stresses should be decreased 20 to 25 per cent., and the minimum thickness of metal increased 25 per cent., unless the metal be fully protected by an acid-proof coating. The best paints at present in use do little more than delay and retard corrosion. The minimum thickness of corrugated sheet steel for roof covering should be No. 20 gage and for the sides of buildings No. 22 gage, but where there is certain to be no danger of corrosion, Nos. 22 and 24 may be used for the roof and sides, respectively.

He states also that the cost of making details for the headworks of mines is from $4 to $6 per ton; for churches and courthouse roofs, having hips and valleys, from $6 to $8 per ton; for ordinary mill buildings from $2 to $4 per ton; the details for all the work done in a year by a large structural concern were contracted at $2.06 per ton, which price netted the contractor a fair profit. The details for the buildings for the Basin & Bay State smelting plant at Basin, Montana, containing 270 tons of steel, cost $2 per ton.

## NOTES ON TIMBER

(June 16, 1904, and July 29, 1905)

The following rules for the inspection of yellow pine, adopted at the recent meetings of the Georgia Interstate Saw Mill Association and the South Carolina Lumber Association, are of interest to constructing engineers: All lumber must be sound, well manufactured, full to size and saw-butted; free from unsound, loose, and hollow knots, worm- and knot-holes, through shakes or round shakes that show on the surface; square edge unless otherwise specified. A "through shake" is defined to be through, or connected from side to side, edge to edge, or side to edge. In the measurement of dressed lumber, the width and thickness before dressing must be taken; less than one inch is to be measured as one inch.

Loren E. Hunt, Government engineer in charge of timber testing for the Pacific coast, in a paper read at the Engineering Congress, at Portland, Oregon, June 29 to July 3, 1905, referred to the first systematic effort to test the mechanical properties of timber, by the late J. B. Johnson, at Washington University, St. Louis, in 1891, and continued by giving in detail the present methods in use by the Bureau of Forestry. He presented data showing comparative results, from large timbers of all grades, to small selected specimens, and from red (Douglas) fir to Western hemlock. The large timber was found to develop a modulus of rupture of from 50 to 90 per cent. of that shown by the small specimens, depending upon the dimensions and conditions of specimens. Western hemlock was found to develop somewhat over 80 per cent. as much strength and elasticity as the red fir. In the discussion, results were given of some recent tests of Alaska spruce, made at the University of Washington, Seattle, in which the results, from 7- by 8-in. specimens, cut from small trees and full of knots, were less (though not to any great extent) than those obtained from Western hemlock and the poorer qualities of fir.

## DESIGN OF TIMBER TRUSSES

(September 26, 1903

The overwhelming importance of steel in modern structural work detracts from the engineering interest in timber work, but the latter is nevertheless still largely used, and doubtless will be so long as it is, for many purposes, cheaper than steel. Steel construction has the great advantage over timber in the fact that the strength of the material can be far more accurately determined. Steel and iron are likely to be fairly uniform; the reverse is the case with timber. It is now generally recognized that the data as to the strength of timber determined from small pieces, as given in the older hand-books and text books, are unsafe. Professor Lanza has concluded, from the results of his numerous tests on large, commercial timbers at the Massachusetts Institute of Technology, that the breaking strength of long-leaf Southern pine should not be figured at more than 5000 lb.; the Division of Forestry of the Department of Agriculture has adopted 4650 lb. as the basis of its calculations. These new data have not upset old calculations, except in so far as they have made it clear that, in assuming maximum fiber stress of 1200 lb., the factor of safety is only about 4, instead of 6 or more, as used to be reckoned.

In the ordinary timber truss, wherein the necessary sizes of the various members have been carefully computed, the weakest points are likely to be the joints. Special care should be given, therefore, to the making of the joints as strong as the other parts. This is emphasized in the valuable little book on the "Design of Simple Roof Trusses" recently published by Prof. Malverd A. Howe, which may profitably be studied by the designers of such trusses. Another valuable reference is a paper by Major G. K. Scott-Moncrieff on tests of full-sized timber trusses, published in the *Journal* of the Royal Institute of British Architects, Jan. 14, 1899. These tests were made largely to determine the accuracy of Tredgold's rules as to the proportions and construction of

timber trusses, on which British practice has been extensively based, and also to show the actual behavior under a breaking strain of more rationally designed trusses. Various of the latter failed, not by the yielding of any member, but by the shearing of the end of the tie beam because of the defective character of the joint. The tendency to lateral deflection was observed to a marked extent, demonstrating the importance of a well-designed lateral bracing between trusses, which function in a roof is to a considerable degree fulfilled by the purlins. The importance of providing means for tightening the various joints at will was also remarked.

The various conditions for a well-designed roof truss of wood are met by the selection of a suitable and economical form; the proper concentration of strains of triangles and parallelograms of forces, avoiding the introduction of bending moments; the accurate determination of the sizes of the various members, using a safe fiber stress as determined by the recent investigations of different timbers, making due allowance for the condition of the latter, and the intelligent arrangement of the joints. In the last connection, little reliance is to be placed on the expensive and complicated methods of mortising and tenoning prescribed in the older methods of carpentry, but rather on the proper use of iron straps, bolts and washers, while cast-iron parts in the form of brace and rafter shoes, king-heads and queen-heads, etc., are often to be recommended.

## SAW-TOOTHED ROOF CONSTRUCTION

(February 3, 1906)

The new erecting and machine shop of the Pittsburg & Lake Erie Railroad at McKees Rocks, Pa., has a modified form of saw-toothed roof that presents some novel methods of dealing with ordinary conditions. The steel work for this building was designed by Albert Lucius, of New York, and was fabricated and erected by the McClintic-Marshall Construction Co., Pittsburg, Pa.

The steel skeleton is enclosed by 18-in. curtain walls of brick; both steel and brick work are carried on concrete foundations. The building is 533 ft. long and is divided transversely into three spans and longitudinally into bays of 22 ft. The erecting shop occupies the higher portion on the south side of the building (a span of 68 ft. 9 in.) and extends the full length, while the machine shop occupies the entire north side; it is approximately 100 ft. wide (in two spans) and also extends the full length.

One electric crane of 120 tons capacity and one of 10 tons capacity, each traveling on a separate runway, serve the erecting shop; their respective spans are 65 ft. and 62 ft. center to center of crane rails; and the machine shop is covered by two 7½-ton cranes, one in each span. The span of these cranes is 46 ft. 3 in. All cranes run the entire length of the building.

The arrangement of the building is shown in the accompanying cross-section, Fig. 1. In the plane of the roof trusses is a system of longitudinal bracing which, in connection with similar vertical bracing between the columns, is designed to take the stresses arising from the load and motion of the crane. The columns which support the roof are separate from the crane columns, but both are composed of channels and plates, of box section; the crane runways are of the usual plate-girder construction, except that lateral stiffness is provided to resist the thrust of the crane when the trolley is suddenly stopped. The girders for the 120-ton crane are 5 ft. deep.

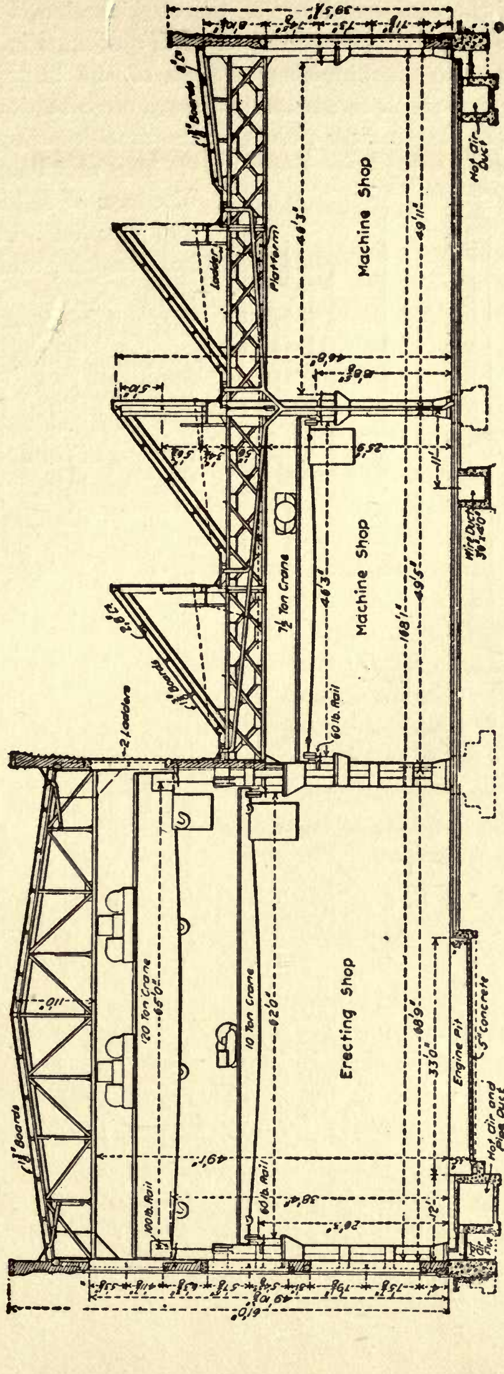



FIG. 1. — Cross-section of Building.

Details of the roof are shown in the cross-section, Fig. 1, and also in the detail of the saw-toothed construction, Fig. 2. This is used above the machine-shop section of the building. The details of the saw-tooth construction show a cross-section through the windows at the quarter points of the machine-shop roof.

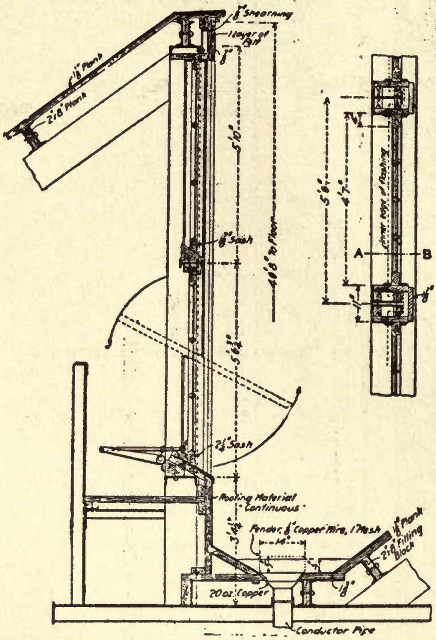


FIG. 2. — Detail of Saw-tooth Construction.

The roof itself consists of a base of 1⅝-in. tongued and grooved boards, upon both erecting and machine-shop sections, and upon this is laid the roofing felt.

Inside drainage is used to eliminate the possibility of water backing up, due to freezing of conductor pipes. The conductor pipes are 5 in. in diameter, on the erecting-shop section, and lead from the flashing boxes to the longitudinal discharge pipes, of which there is one 8 in. in diameter, carried along under each side of the roof and supported by the steel work, as shown in Fig. 1. For the machine shop, 4-in. conductors lead from the flashing boxes to 5-in. vertical discharge pipes, which are arranged alongside of alternate columns of the middle row, extending through the shop. There are 12 low points in the roof construction to provide drainage.

The saw-tooth windows are arranged to be operated in sections from the floor of the shop by hand-wheels; the mechanism for this purpose is shown in Fig 2. This gives ample ventilation, and obviates the necessity for any further provision for that purpose.

## NOTES ON ROOFS AND ROOF COVERINGS

By W. R. Ingalls

(September 5, 1903)

There is a great variety of roofing material, but for mining and metallurgical works it is necessary to obtain a roof which will be reasonably cheap and at the same time fairly durable. The element of first cost excludes roofing with slate and the more expensive metals, which for general architectural work are doubtless the best; moreover, the acid fumes given off in many metallurgical works would be very destructive to certain metals, such as sheet zinc and copper. With these limitations, the question of roofing is often a serious problem, unless it be possible to design the buildings so that the roofs will have such a slight pitch as 1:24, commonly called a flat roof, in which case the problem is simplified by putting on a covering of tar and gravel, which makes not only one of the best kinds of roof in respect to durability, low cost of maintenance and safety against fire, but also is one of the cheapest; in fact, it is probably the cheapest of first-class roofs. The limitation to the use of tar and gravel, however, is the inability to apply it properly to roofs of steep pitch, the tar having a tendency to melt and run off in the hot weather of summer, and unfortunately the exigencies of metallurgical construction demand often a roof of 30 deg. to 45 deg. pitch. It is the most serious question, therefore, to determine what may best be put on such a roof.

It may be assumed that such a roof will be trussed, the trusses bearing the purlins to support the roof sheathing. The roof sheathing may consist of metal nailed directly to the purlins; or it may be some flexible material laid on 1-in. boards, the latter being nailed to the purlins so that they will be parallel with the pitch of the roof, or else nailed to jack rafters which are themselves carried by the purlins; of these two methods, the former is the cheaper; the stiffness of a yellow pine board one inch thick is

such that, when nailed to purlins spaced 36 in. apart, center to center, the roof will be good for a snow and wind load of 40 lb. per square foot, which is generally assumed in designing in the northern part of the United States.

Roofs prepared in the manner described above may be covered with corrugated iron, usually of No. 22 gage, laid directly on the purlins. If the purlins be boarded over, the boarding may be covered with corrugated iron of a lighter gage, say No. 27, or with shingles, or with some kind of so-called ready roofing. The latter material is usually a woolen felt saturated with coal tar and rolled out in two or three layers, with a layer of coal tar composition between the layers of felt. If the material consists of two layers of felt, it is referred to as two-ply; if of three layers, it is referred to as three-ply. Similar ready roofing is made of felt saturated with asphaltum, or with paraffine, or some other analogous waterproof substance.

The ready roofing is put up in rolls of convenient size. In laying on the roof, a roll is unwound parallel with the eaves, and after nailing to the boards another roll is unwound and laid so as to overlap the first by about two inches. The lap is cemented by means of a composition of pitch, and is securely tacked down. After the entire roof is covered, the tack heads at least, and sometimes the whole roof, is coated with composition, which may finally be sprinkled with sand or fine gravel. The sand or gravel greatly increases the durability of the roof. A roof laid with good three-ply material, the laps well cemented together and tacked down, and the surface covered with coal tar, using about 1.5 gals. per 100 sq. ft., which is about as much as can be made to remain on a roof of steep pitch, and finally thoroughly sanded, makes about as good a roof as is required for ordinary purposes. Its cost should not be more than $2 to $2.50 per square, completed.

A shingle roof is durable, but is somewhat more expensive than a roof covered with prepared felt. At present prices corrugated iron is one of the most expensive of the cheaper roofing materials, and also it is one of the least durable, its first cost being increased by the necessity for repainting the sheets as they are received from the factory, and the cost of maintenance being considerable because of the further paintings that are frequently required if the life of the material is to be preserved at all.

The approximate cost of various roofing per 100 sq. ft. is summarized, as follows:

| | |
|---|---|
| Boards and tarred felt: | |
| 110 ft. of 1-in. board at $20 | $2.20 |
| Carpenters' labor, nails, etc. | .50 |
| Three-ply ready roofing, in place | 2.50 |
| Total | $5.20 |
| Boards and shingles, laid 4 in. to weather: | |
| 110 ft. 1-in. board, at $20 | $2.20 |
| 900 shingles, at $3 per M | 2.70 |
| Roofing paper | .25 |
| Labor, nails, etc. | 1.91 |
| Total | $7.06 |
| Corrugated iron No. 22 gage, laid directly on the purlins: | |
| 160 lb. of corrugated iron sheets, at 3c | $4.80 |
| Labor, nails, etc. | 1.00 |
| Painting 214 sq. ft. | 1.41 |
| Total | $7.21 |
| Corrugated iron, No. 27 gage, laid on boards: | |
| 110 ft. of 1-in. board, at $20 | $2.20 |
| 89 lb. of corrugated iron sheets, at 3.6c | 3.20 |
| Roofing paper | .25 |
| Labor, nails, etc. | 1.50 |
| Painting 214 sq. ft. | 1.41 |
| Total | $8.56 |

In the above estimates the price of corrugated iron has been reckoned at 2.4c. per lb. for No. 22 gage and 3c. per lb. for No. 27 gage in car-load lots f. o. b. Pittsburg, Pa., or Youngstown, O. It has been reckoned that the under side of the sheets is to be painted with silica-graphite before the sheets are put on, and that the entire upper surface is to be painted with the same. In the case of the corrugated iron laid on boards, and also in the case of the shingle roof, it is assumed that a sheet of good waterproof paper will be interposed between the boards and the iron or the shingles, respectively. In computing the cost of the boards allowance of 10 per cent. is made for waste. It is assumed that the boards will be laid on purlins spaced 36 in. apart center to center. In the case of the corrugated iron of No. 22 gage,

the purlins may be spaced a little further apart, and there will be a small saving on that account, but this will not be of great significance.

Tar and gravel covering is generally put on a roof of 1-in. boards, laid on rafters, or else on 3-in. plank, grooved and splined (or tongued and grooved), laid directly on the main roof girders. The latter makes the better roof, and is prescribed by the insurance companies for standard mill construction. In that form of construction the roof girders will be heavy beams, spaced something like 8 to 10 ft. apart; if the beams be spaced 10 ft. apart, they should be not less than 8 by 10 in. for a 16-ft. span. If the spacing be more than 10 ft., which is unusual, a heavier plank than 3-in. is required. A 3-in. plank on 10-ft. span is good for a net load of 40 lb. per sq. ft. within the limit of deflection that can be allowed.

In making a tar and gravel roof, several layers of saturated woolen felt are first put on. The surface of the roof is finally covered with coal-tar pitch, and a sufficient body of well screened gravel to give the desired surface. If four layers of felt are put on, the roof is referred to as four-ply; if five layers, it is referred to as five-ply.

There are various methods of putting on the felt and pitch. One method for a five-ply roof is to put on a first layer of heavy dry felt, with 2-in. lap. The other four layers are then put on in courses parallel with the eaves, each layer lapping the one below by 27 in., so that the roof will be five layers in thickness over all its parts. Each layer is well mopped with hot pitch for a distance of 9 in. from the edge. The felt is secured to the roof by 3-dwt. nails with tin disks, driven in rows 10 ft. apart and 12 in. apart in the rows. The entire surface of felt is finally coated with straight run coal-tar pitch and covered immediately with a sufficient body of well screened gravel.

Another method is to put on three layers of saturated felt, as described above, each layer lapping 24 in.; then cover with pitch and apply one or two layers of felt separately in hot pitch. This makes a very good and durable roof.

A third method, which is the one most commonly employed, at least in the Eastern States, consists in putting on three layers of felt over the whole surface, lapping each layer 3 in., and then mopping thoroughly with pitch, using not less than 3 gal. per

square. Two more layers are then laid in hot pitch, so that the roof will have five plies of felt over all its parts. The last layer is secured to the roof at the laps by 3-dwt. nails with tin disks spaced 30 in. apart. The surface of the roof is finally covered with hot pitch, using not less than 8 gal. per square.

In all methods the roof is finished against chimneys, party walls, scuttles, etc., by turning up the felt 4 in. against the wall, and over this laying an 8-in. strip of felt with half its width on the roof. The upper edge of the strip and the several layers of felt are fastened to the wall by laths or battens securely nailed, and the strip of felt is pressed into an angle of the wall and cemented to the roof with hot pitch, the lower edge of the strip being nailed to the roof every 4 or 5 in. Especial care must be taken in fitting around the angles of chimneys and skylights.

The felt for a tar and gravel roof should be made of the best woolen rag fiber, and should weigh about 15 lb. per 100 sq. ft. of roof. The felt comes in rolls, containing 108 sq. ft. The coating composition should be straight run coal-tar pitch; it should be the residuum from a distillation at comparatively low temperature, distillation at too high a temperature being liable to drive off some of the valuable constituents. The pitch will weigh about 10½ lb. per gal., and not less than 11 gal. should be used. The gravel should be clean and free from loam, and should be as dry as possible. If the roof is laid in cold weather the gravel must be applied hot. The gravel is used in size varying from that of a pea to about ½-in. diameter. Sufficient gravel is required to cover the roof thoroughly, about four bushels per square being used generally in laying a good roof. The cost of putting on a first-class five-ply tar and gravel roof is approximately as follows:

| | |
|---|---|
| 75 lbs. of felt, at 3c. | $2.25 |
| 11 gals. of pitch, at 11c. | 1.21 |
| 4 bushels of gravel, at 20c. | .80 |
| Nails, tin disks, etc. | .10 |
| Labor | 1.00 |
| Total | $5.36 |

The cost will, of course, vary according to the prices for various materials, especially the felt, which is worth generally 2c. to 2¾c. f. o. b. factory.

Tar and gravel roofs are frequently put on for less cost than \$5 per square; even for as little as \$3 per square; these cheaper roofs are generally of only four-ply, however, and the material used is neither first-class in quality, nor in sufficient quantity. The felt will be of light weight, and the quantity of pitch employed will not be what it should be.

The putting on of tar and gravel roofing should be entrusted only to experienced roofers. Specifications for such roofs should prescribe the number of layers of felt to be used, its weight per square, and the quantity of pitch and gravel. A good method in cases where a large amount of roofing is to be done is to buy the material directly from the dealers and contract the labor. The builder may then be sure what quality and quantity of material is being used.

## PROTECTION OF IRON AND STEEL

(September 5, 1903)

Maximilian Toch in the *Journal of the American Chemical Society* for July, 1903, discusses the question of the permanent protection of iron and steel, which is one of the greatest importance, in view of the large extent to which steel is employed in modern construction. Both exposed and imbedded iron are subject to progressive oxidation under certain conditions. Manufacturers of paints have long endeavored to devise a coating of material which will prevent corrosion and oxidation. The success of such coatings depends chiefly upon the skill of the workman and their proper application. A good paint improperly applied is relatively as poor as a paint of less merit. Red lead, for example, is condemned as often as commended; it is probable that those who have commended it have had it properly applied by skilful workmen under favorable conditions, and then have had it covered by better paint.

It is a blunder to apply a corrosive oxide to a material that will corrode. If the paint itself be a carrier of oxygen and the iron or steel be subjected to the action of alternate dampness or dryness or of air charged with carbon dioxide, progressive oxidation is sure to take place and the tensile strength of the metal to be materially reduced. Such conditions may readily occur when a beam is placed in a porous wall. If a clean, pure cement concrete is packed hard against an iron or steel surface, little or no oxidation can take place, especially if free lime has been liberated in the setting of the cement, but violent oxidation may take place if cinder concrete containing iron oxides, other metallic oxides, free chlorine, or any trace of a sulphide be used. Pieces of anchor chains imbedded in concrete more than 200 years have been found in Spain in a state of perfect preservation. Large quantities of metal unearthed in Italy and Greece, extremely old, imbedded in cement or concrete are wonderfully well preserved. These observations caused Mr. Toch to experiment with portland

cement for the protection of iron and steel. The result of his experiments led him to conclude as follows:

1. If a proper cement paint be applied to a surface which has begun to oxidize, further oxidation will be arrested.

2. If the cement used be very fine and free from iron, calcium sulphate, and sulphides of low specific gravity, it will quickly set on the surface and eventually become thoroughly fixed upon the metal so that rain will not wash it off.

3. When thoroughly applied, even to three coats, the concrete may be painted with alkali proof and adherent paint, affording absolute protection to iron, so that moisture, carbon dioxide, or factory fumes will not penetrate.

4. Cement paste for application to iron or steel must be made with pure water, and the mixture must be stirred at least fifteen minutes to admit of the liberation of the lime.

5. Free lime on the surface of the cement coating is quickly carbonated, and then has no injurious action upon linseed oil paint, which may, under such conditions, be applied and become extremely efficacious.

It is reasonable to believe that structural metal work coated with a layer of cement paint and further protected by a layer of hydrocarbon insulating paint, when imbedded in masonry, will be perfectly immune to oxidation and probably will last for all time. A similar coating will afford sufficient protection to pipes and conduits placed in the ground and subjected to various influences, such as of moist gases, electric currents, acid and alkaline liquids. Pure portland cement mixed with water cannot be used as a metal wash because it will not always set and is apt to crack when it does; hence it must be diluted and care must be exercised so as not to impair the strength of the cement. Voids can be prevented by careful brushing, and for certain structural work, where application by the brush is inapplicable, spraying is effective, but the cement must then be applied in several layers.

The waterproofing of brick walls from the outside as a protection against penetrating rain or dampness is an important consideration. A newly laid brick contains as much as 8 oz. of water, and its power to adhere to the mortar increases with the quantity of water it contains. If a linseed oil paint be applied to a newly erected and wet wall it quickly peels off and ruins the wall for the further application of paint. However, if a proper

cement mixture be applied to such a wall in the form of a paint or wash it not only adheres perfectly, but forms an excellent base for the application of a good linseed oil paint. Painting the outside wall of a building in this way is something of a protection to the iron and steel used in construction, since it prevents to a large extent the access of carbon dioxide, moisture, and gases.

## PROTECTION OF STEEL FROM CORROSION

By Charles L. Norton

(February 18, 1904)

Previous tests, carried out on perfectly clean steel, have been repeated on specimens in all degrees of initial corrosion, doubt having existed as to whether the results found with clean steel would apply to rusty or dirty steel. The specimens were imbedded in concrete mixed in the proportion of 1: 2.5: 5 to 1: 3: 6. The cements used were Alpha, Lehigh, and Alsen. Care was taken in selecting the sand and stone, the latter being of such size as to pass a 1-in. mesh.

The results of the tests, which were carried out under various conditions, lead to the conclusion that structural steel, if encased in a sound sheet of good concrete, is safe from corrosion for a very long period — longer than the changes in our cities will allow any building to remain. It is necessary, however, to be sure that the steel is properly encased in the concrete, and, because of the difficulty in getting sound work, many engineers will not use concrete. This is especially true of cinder concrete, in which the porous nature of the cinder has led to much dry concrete, and many voids and much corrosion.

There can be no question that cinder concrete has rusted great quantities of steel, not because of its sulphur content, the danger from which is a myth, but because it was mixed too dry, through the action of the cinders in absorbing moisture, and therefore contained voids; and, moreover, because the cinder often contains oxide of iron, which, when not coated with the cement by thorough wet-mixing, causes rusting of the steel which it touches. If cinder concrete be mixed wet and mixed well, it may be trusted as much as stone concrete, so far as corrosion is concerned.

## SPECIFICATIONS FOR PAINTING STEEL STRUCTURAL WORK

**(October 10, 1903)**

The painting specifications for the superstructure of the Blackwell's Island bridge across the East River at New York, contracts for which have just been let, are of general interest to engineers. They are as follows:

All material shall be received and painted under cover; and no painting, either at the works or in the field, shall be done in wet or freezing weather. All structural steel and iron before leaving the shop must be thoroughly cleaned of all mill scale, dirt, and rust, by the use of small hammers, steel scrapers, and wire brushes, and of oil by the use of benzine, and must then receive one coat of red lead and boiled linseed oil. These materials must be brought to the work in their original packages and mixed in a revolving churn just before using, in the proportion of 33 lb. of dry red lead to one gallon of linseed oil.

The red lead must be strictly pure, and shall contain at least 80 per cent. of true red lead (of the composition $Pb_3O_4$); the total amount of lead present shall not be less than 89 per cent., of which not more than 0.1 per cent. shall be present as metallic lead. The color shall be clean and pure tint. The red lead shall be of the fineness that, when washed with water through a No. 19 silk bolting-cloth, no more than 1 per cent. shall be left on the screen. The linseed oil shall be an absolutely pure boiled oil, containing no matters volatile at 212 deg. F. in a current of hydrogen; it shall not contain any rosin or manganese. The oil shall be perfectly clear on receipt and no deposit shall form on standing, provided the oil is kept at a temperature above 45 deg. F. The film left after flowing the oil over the glass and allowing it to drain in a vertical position must be dry to the touch after 24 hours. All finished surfaces shall be coated with white lead and tallow before being shipped from the shop.

After the structure is in place all mud and dirt that may have accumulated during the erection must be removed, and all

abrasions in the first coat of paint must be thoroughly brushed with a stiff wire brush and such places "touched up," and all bolt heads and location marks thoroughly painted with the paint, as described, and then all the steel and iron work shall be thoroughly and evenly painted with an additional coat of red lead and boiled linseed oil, of the quality stated, mixed in a revolving churn in the proportion of 33 lb. of red lead to one gallon of oil, with the addition of $\frac{1}{2}$ lb. of best lampblack (ground in oil) to every 99 lb. of red lead used. The second field coat shall consist of red lead and boiled linseed oil, without lampblack.

## STAMP MILL CONSTRUCTION

(February 23, 1905)

In replying to the discussion of his paper on "Mill Construction, Milling, and Amalgamation," I. Roskelly stated (*Journal* of the Chemical, Metallurgical, and Mining Society of South Africa) that many points of agreement among mill-men had been brought out, among which should be emphasized:

1. More mill space than is usually allowed should be provided.
2. Concentrates should be recrushed.
3. Special precautions should be taken for reducing vibration.
4. Solid wooden guides have yet to be improved upon.
5. Extra mechanical contrivances for facilitating work should be supplied.
6. Special precautions should be taken if cyanide is used.
7. Outside amalgamation is to be preferred.
8. A thoroughly adequate crusher plant should be provided.
9. Ten stamps on a shaft is not an improvement.
10. The new Blanton cam is an improvement on keyed cams.

The so-called improved feeders do not, in Mr. Roskelly's opinion, justify discarding the Challenge feeder, but probably an improvement will be made in that, so that the brake-wheel and three pawls can be done away with.

Most contributors to the discussion advocated either the sparing use of cyanide or its total abolition. Mr. Roskelly considers, however, that it merely cleans and brightens the plates and has never found that it hardens amalgam, nor does it dissolve amalgam to any appreciable extent.

Outside amalgamation pure and simple has proved itself to be as good as, if not better than, any other method, and it is the method which reduces losses and risks of all kinds to the minimum. With inside amalgamation the mill-man is working more or less in the dark.

## DESIGN OF ORE-BINS AND COAL-POCKETS

(May 5, 1904)

This is a subject that is apt to be troublesome to the metallurgical engineer, because of the lack of data. The gross weight that must be carried is easily estimated, but the manner in which it will exert its pressure, not so easily. Recourse has been made to the formulas for retaining walls and other indirect methods. S. A. Jamieson, elevator engineer, of Montreal, Can., in a paper on "Grain Pressures in Deep Bins," read before the Canadian Society of Civil Engineers, December, 1903, has thrown some light on this subject. Although his experiments referred especially to the behavior of cereal grains, he also showed that dry sand acted in a quite similar way, and it is not improbable that the same general rules apply to heavier mineral particles under the same conditions of dryness, etc.

Mr. Jamieson's paper is too long and intricate to be summarized satisfactorily in a brief note. The nature of his conclusions, however, is indicated in the statement that, in a deep bin, only a small proportion of the weight of the material is carried on the bottom, the major portion — depending upon the depth of the bin — being exerted against the sides, where it is resolved into vertical pressure by friction. The proportion of the total weight of grain in a bin that is carried by the walls and on the bottom, and therefore the intensity of both the vertical and lateral pressures, is entirely dependent upon the following factors: (1) Coefficient of friction between the granular material and the bin walls; (2) ratio of the breadth or diameter of the bin to the depth; (3) ratio of the horizontal area or weight of the column to the area of the bin walls; (4) angle of repose of the granular material, or the ratio of lateral to vertical pressure.

## NEW CHANGING-HOUSE AT CLIFFS SHAFT MINE [1]

BY JOHN S. MENNIE

(September 12, 1903)

On the night of Dec. 1, 1901, the changing-house at the Cliffs Shaft mine was totally destroyed by fire, causing quite a financial loss not only to the Cleveland-Cliffs Iron Company on the building, but also to its employees on their clothing. The losses on buildings of this description, when built of wood, would indicate that they are extra hazardous, and in fact the insurance rates are almost prohibitory. In considering the erection of a new building it was thought that it would be cheaper in the end to build it of brick, which, while not absolutely fireproof, would be practically so. The other essential requirements were an easy method of keeping the building clean and the best possible arrangements for the comfort and cleanliness of the occupants.

The building as erected is 30 ft. 4 in. in outside dimensions and 11 ft. from the top of the floor to the under side of trusses. The foundation walls up to the floor line are built of common rubble stone, laid up in cement. The exterior walls above the floor line are of common brick laid up in lime mortar, and are 10 in. in thickness, consisting of two 4-in. courses of brick with a 2-in. air space to prevent sweating of walls. The two courses of brick are tied together every fifth course with clipped headers. Where the trusses rest on the walls piers are formed by building an extra course of brick on the outside. Three division walls run across the building; they are solid, 8 in. thick, and are carried up to the roof boards as fire walls.

The roof is carried on light trusses of 6- by 6-in. timber, spaced about 8 ft. centers. On these are laid 2-in. matched and dressed common pine, face down; strips 1 by 2 in. in size are then nailed on across this planking, spaced 2 ft. centers and then covered

[1] Abstract from *Proceedings* of Lake Superior Mining Institute, August, 1903.

with 1-in. common hemlock shiplap, making a 1-in. air space in the roof. The roof covering is two-ply rubberoid roofing. The exterior woodwork is painted two coats of reddish brown paint, the interior woodwork two coats light drab paint, and the interior brick walls were given one coat of white cold water paint.

The floors throughout are of concrete and all are graded to the center of the rooms where connection is made through gratings to a 6-in. tile sewer-pipe. These pipes are then carried under the floor to the outside of the foundation walls and empty into a box drain. The floor under the shower baths is graded to the wall to a drain of sewer-pipe split in halves, and these are connected to the main drain.

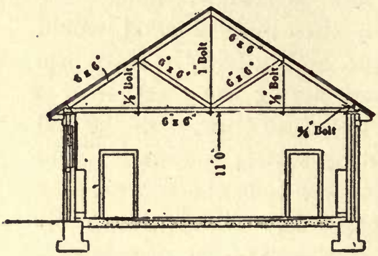


Fig. 3. — Cross-section.

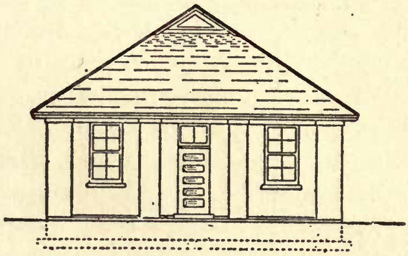


Fig. 4. — East Elevation.

The entrance to the building is by two doors, one at each end of what is termed the change-room. This room is 28 by 54 ft. in size. Against the walls of this room are set up 126 lockers 12 by 12 in. by 5 ft. high, for street clothes. They are made of expanded metal, and being each used by a night and a day man, they accommodate 252 men.

In the center of the room are two drying racks for mine clothes. These are made from 1-in. iron pipe and fittings and have four bars each. Over each of these racks is a large ventilating hood made of galvanized iron on $1\frac{1}{4}$-in. angle-iron frames; these rise into three 16-in. pipes which are carried through the roof and capped with 16-in. "Star" ventilators. Stationary benches of 2-in. plank on iron standards extend the full length on each side of the room between the lockers and the drying racks.

From the north end of this room entrance is had through two doors to the wash-room, which is 28 by 46 ft. in size. Running lengthwise of this room are three wash-troughs. These are

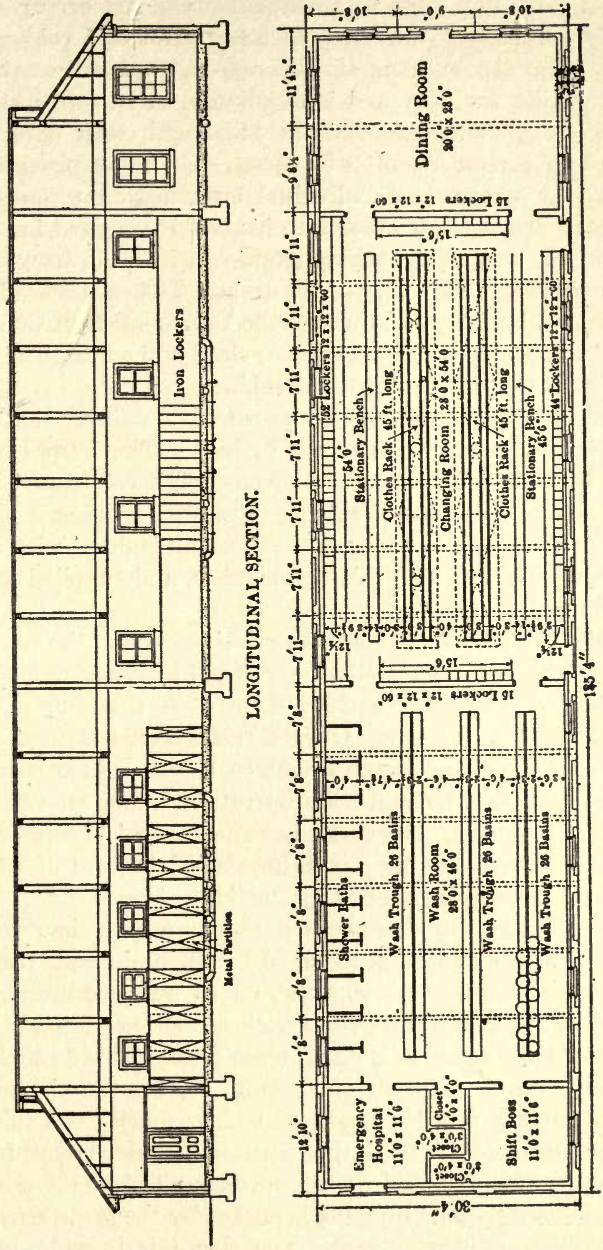



FIG. 5. — Floor Plan.

made of $\frac{1}{8}$-in. iron bent to a semicircle and are set up on iron standards built into the concrete floor. Hot and cold water is brought into the building through separate pipes in a tile-pipe conduit under the floor and is distributed in two separate pipes to each trough and extended the full length with upright connections to faucets about 5 ft. apart. On these pipes and the edge of the trough rest individual enameled iron wash-basins. Each man draws clean water into his basin tempered for heat to suit himself, and after washing empties it into the trough, when it immediately runs off to the drain. This system of wash-troughs and basins was in use at the Champion Iron Company's mine at Beacon, and seemed to be so clean and admirable in every way that it was adopted in this building.

Against one of the walls are arranged ten shower-bath stalls about 4 ft. square in size and 7 ft. high. The frames of these are of 1-in. angle iron and are covered with corrugated iron to within about 18 in. from the floor. The door opening is provided with a drop curtain of linoleum. Each stall is fitted with a spray shower-bath fixture with mixing chamber, and supplied with hot and cold water.

In the north wall of this room are three doors, the center one opening into a janitor's closet for keeping brooms, hose, etc., the other two into rooms each 11 ft. by 11 ft. 6 in. in size. Both rooms have large closets. One of these rooms is used by the shift-bosses as a change-room and office, the other as an emergency hospital, and is fitted up with operating table, stretcher, and surgical appliances. The closet has shelves and is supplied with such medical supplies as might be needed in case of accident. This room has a wide door as an outside entrance.

Returning to the change-room there are two doors opening from its south end into a room 20 by 28 ft. in size, used as a dining-room. This room was put on as an experiment. It is fitted with two long tables and benches. Being a separate room and separately heated it is much more comfortable to sit in than the change-room. Its use also tends to keep the change-room cleaner. As the room is now used to its capacity it would seem to be advisable to extend this feature in any similar building hereafter erected, so as to accommodate all the men who may use the building. The building is heated by the Webster vacuum system, using exhaust steam. The steam is brought into the

building under the floor through iron pipes in a tile-pipe conduit and distributed to the different coils of pipe. These coils are placed under the lockers, benches, clothes-racks and wash-troughs. The other rooms are heated by wall coils.

The entire cost of the building was $6604.

Commenting on the building after being in use, I would say that the lockers for street clothes are rather small and that the 12 by 16 in. locker of same make should be used. They would be large enough in cases where the one shift is up and changed before the other shift begins to change. The amount of wash-trough room is more than is required, and might be cut down one-third for the number of men arranged for in the building. The danger of fire is reduced to a minimum, and in fact is practically confined to the clothes in the change-room. The doors in this room leading to other rooms have been tin-clad, and we think, if a fire should occur in the clothes, that no part of the building would be damaged to any material extent.

## DUST-PROOF PARTITIONS

(October 3, 1903)

In a dry-crushing mill, there is sometimes great difficulty in constructing partitions which will absolutely confine the dust. No matter how carefully matched boards are put up, or paper sheathing is put on, some dust will work through.

A good and cheap method of preventing this is to make the partition of ordinary boards and seal the joints with strips of muslin, glued on. Cut the muslin into strips about 4 in. wide. Apply glue to the boards for about 1 in. on each side of the joint. Lay on the strip of muslin and with a case-knife (or similar thin, blunt tool) crease it into the joint. Be sure that the muslin is thoroughly glued to the boards. The crease of the muslin inside of the joint allows for the opening of the latter as the boards shrink, and the crack remains sealed by a medium which will prevent any dust from passing through.

This method is due to the late Henry A. Vezin, to whom the mining and metallurgical profession owes many useful inventions. His method of making dust-proof partitions has been adopted in many dry-crushing mills, and it has proved thoroughly serviceable.

# PART III

# ORE-CRUSHING MACHINERY

## CAPACITY OF BLAKE CRUSHERS

By W. R. Ingalls

(October 31, 1903)

The capacity of a crusher of the Blake type depends chiefly upon the width of the jaw, the speed at which it is driven, the size to which the ore or stone is crushed, and the character of the ore or stone. As in the case of rolls and other crushing machines, the capacity is more accurately stated in terms of volume, i.e., cubic yards or cubic feet, than in terms of weight, because, all other things being equal, a crusher will deliver more tons of a heavy ore than of a light ore. With respect to the character of the material to be crushed, hard ore or stone that breaks with a snap will go through faster than less brittle stuff; one of the hardest materials to crush is soft, talcose, tough stuff, that mashes and is broken only with difficulty. With some such kinds of material it is almost impossible to get it through the crusher at all.

There are practically no published data as to the capacity of crushers on different kinds of ore, nor of the power required. The data given in the catalogues of various manufacturers are quite unreliable, varying greatly. As a matter of interest I have summarized the statements with respect to the 9- by 15-in machine given in the catalogues of eight prominent manufacturers. The following table gives the total weight of the machine manufactured by each, the speed at which it is recommended to drive, the estimated horse-power required, and the tons of product made per hour in crushing to about 1.5-in. size. Many of these manufacturers state that the figures quoted are only approximate, but few express any recognition of the fact that output should be stated in cubic feet and not in tons, and few specify as to the character of ore upon which their figures are based. The presumption is that quartzose ore is especially referred to.

DATA OF 9-IN. BY 15-IN. BLAKE CRUSHERS

| | WEIGHT, LBS. | SPEED, R. P. M. | H. P. REQUIRED | TONS PRODUCT PER HOUR |
|---|---|---|---|---|
| No. 1 | 16,000 | 250 | 10 | 15.0 |
| No. 2 | 15,000 | 250 | 12 | 5.5*b* |
| No. 3 | 16,900 | 250 | 10 | 10.5 |
| No. 4 | 15,750 | 300 | 12 | 5.5*c* |
| No. 5 | 14,000 | 275 | 15 | 11.0*f* |
| No. 6*a* | 16,500 | 275 | 12 | 16.0*d* |
| No. 7 | 16,500 | 250 | 10 | 8.0*e* |
| No. 8 | 16,000 | 250 | 10 | 10.5*f* |

(*a*) The crusher made by this manufacturer is 10 in. by 16 in.

(*b*) This manufacturer rates the capacity as 10 tons per hour in crushing to 2.5-in. size, 8 tons in crushing to 2-in. size, and 5.5 tons in crushing to 1.5-in. size.

(*c*) This manufacturer gives the same figures as the one quoted above, with the additional statement that in crushing to 1-in. size the capacity is only 4 tons per hour.

(*d*) The size of the crushed product is not stated by this manufacturer.

(*e*) This manufacturer rates the capacity as 8 tons per hour in crushing to 2-in. size, 4 tons in crushing to 1-in. size, and 2 tons in crushing to 0.5-in. size.

(*f*) This figure is for crushing to 2-in. size.

The machines to which the data here tabulated refer are built by well-known and reputable firms, whose work has stood the test of years. It will be observed that there is very little difference in the quantity of material put in any of them, the weights running in all cases close to 16,000 lb. There is a close agreement in the idea that the proper speed for these machines is 250 to 275 revolutions per minute, and with a single exception the estimate of power required is 10 to 12 horse-power. With respect to the rated capacity, however, there is a wide variation, this ranging from 5.5 tons per hour crushed to 1.5-in. size up to 15 tons per hour crushed to the same size. It is obviously impossible that two equally well-built crushers of the same size and type should show such differences as 0.66 horse-power and 2 horse-power per ton of stone broken (in all well-built crushers the friction is about the same), and the reporting of such data betrays ignorance somewhere. There is no doubt that the manufacturers who report such an hourly capacity as 15 tons greatly overestimate the actual efficiency of their machines. One

well-known manufacturer, who treats of the subject more carefully than most, states that in crushing "ordinary material," with the jaws of the crusher set about 2 in. apart when closed, from 1.25 to 1.5 horse-power per ton of material crushed is usually required.

For practical purposes the assumption that a 9-in. by 15-in. Blake crusher, run at 250 revolutions per minute, will break to 1.5-in. size about 5 to 7 tons of brittle quartzose ore per hour with the consumption of 15 horse-power is probably sufficiently safe. The actual duty will be governed largely by the manner in which the crusher is fed, the proportion of fine ore in the stuff crushed, and upon whether or not the fines are screened out before feeding to the crusher. However, it is to be regretted that there is such a lack of accurate data as to these common machines, and especially such a lack of data as to the comparative hardness and toughness, breaking strength, friability and specific gravity of various ores. Such data with respect to the typical ores would be of great value to the metallurgist.

# A CANTILEVER BATTERY FRAME

By Ira C. Boss

(March 10, 1904)

The invention of the gravity stamp has been credited to Von Maltitz about the year 1505, but the first commercial use of it was made by Paul Gromstetter in 1519, when he established, at Joachimstahl, a process of wet stamping and sifting. Until the nineteenth century the stamp was merely a square timber (with the exception of a few instances when square iron was used) with an iron shoe at the bottom. With the discovery of gold in California came the introduction of this crude machine, and rapid improvements in its construction soon followed.

C. P. Stanford introduced the round iron stem, and Isaac Fish suggested a method of revolving it. Zenos Wheeler and H. B. Angel together invented the method of holding the tappet by means of a gib and two cross-keys. Irving M. Scott designed and constructed the first double-arm cam. Zenos Wheeler is also credited with the high-box mortar. About this time the long battery-blocks set on end came into use, but the originator is not known. A general perfecting of details still goes on from year to year.

In stamp-mill design much attention has lately been paid to the construction of the mortar. A vigorous effort has been made to render the quadruple discharge popular, but it was handicapped at the start by faulty construction and impatient criticism. The low mortar with long stamp-head is being built both at Chicago and San Francisco. The object of building this low mortar is to make it possible to guide the stem at a much lower point than is generally practised, so as to decrease the vibration of the stem and diminish the progress of crystallization. The guides also should be a snug bore, 1-64 part of an inch on the diameter, affording plenty of clearance. This necessitates a careful alinement and a steady back for the guides. Cast iron

will wear the stem much less than wood. A solid cast-iron guide not less than 12 in. long will give excellent results.

Forged-steel shoes and dies give good service, but are more expensive than the best cast steel and wear no better. The self-tightening cam is being largely used, as it is easily and quickly removed. If the battery frame is rigid there will be less and a more even wear to the stem, shoe, and die, and less power required to do the work. A battery frame of steel is better than one of wood, if constructed so as to have a minimum of vibration, which will insure its own life and that of the stems, by holding the guides firmly and always in alinement. The enormous production of steel and the improved railroad facilities, together with the increase in price of timber, have made it possible in many localities to lay down steel as cheaply as, if not more cheaply than, timber.

The accompanying drawing shows the wooden cantilever battery frame and ore-bin. This is designed for districts where timber is plentiful and when time is limited.

A 10-stamp mill of this design, of steel construction, has just been erected at Tonopah, Nevada. It has foundations and battery frame for 20 stamps, but so far only 10 have been installed. The battery and ore-bin frames, the pan frame, and building are all made of steel, and the foundations and floors are concrete. The object was to do away with the future replacing of timbers, to give guides and bearings rigid support; the cement floors were put in to prevent loss of quicksilver.

The battery frame is a part of the ore-bin from which it gets its rigidity. Two concrete walls 30 in. thick are the foundations for a series of 24-in. I-beams having 5.5-ft. centers. These beams extend out so as to take the bearings for the cam-shaft. The skeleton of the bin is of 10-in. I-beams and the 10-in. cap-channels are a support for a hanging strap, which catches the base beam near the cam-shaft bearing, giving that bearing an extra support. This construction leaves a clear passage around the two five-stamp mortars, which are of the double discharge and low type.

The lower guides are placed so low that, when the stamp is at its highest point with new shoe and a new die, the top of the stamp-head nearly touches the bottom of the guide. The housing is around the stamp-head instead of enclosing the stem; also the stamp-head is reached more handily in "backing off" the head. The sides and bottom of the ore-bin are of reinforced concrete,

arched. The ends of the bin are concrete walls, the bin and battery occupy a minimum of space.

The Tonopah mill consists of ten 1300-lb. stamps; the guides are individual, of cast iron and 15 in. long. The pulp is crushed wet, put through a coarse screen and piped to two 5-ft. Huntington mills, used as regrinders. From these mills it is taken to a series

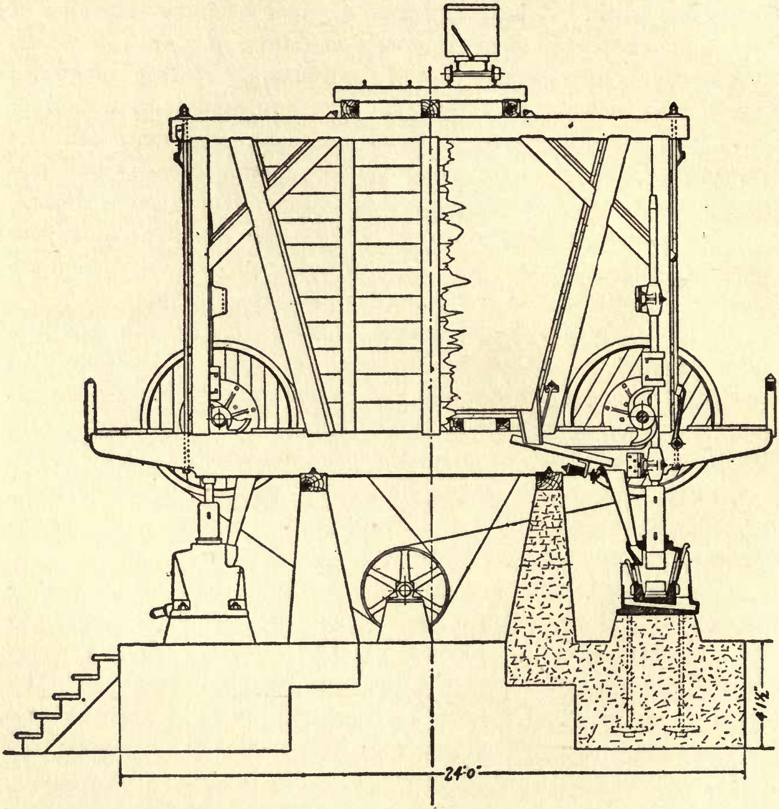



FIG. 6. — Cantilever Battery Frame, Wood Construction.

of continuous pans and settlers. The pan frame is steel and the pans are overhead driven, doing away with the noisy gears which are usually placed in a dark and inconvenient pit.

The cams, tappets, shoes, and dies are of chrome steel. The cams are self-tightening, as is also the cam-shaft pulley. A unique detail of the mill is a contrivance for hanging up the stems.

Cast-iron fingers with adjustable steel caps are mounted on a 4-in. shaft; a lever for each finger passes between the stems and projects in a handy position, so that, by simply pulling down the lever with one hand, the tappet is lifted out of the way of the cam. Lifting the lever puts it at work again, and it is all done from in front of the mortar, where one wants it.

The concrete foundation under the battery is 6 ft. deep and 7 ft. wide. The proportion of cement and sand was varied. The first 2 ft. were mixed, 6 of sand to 1 of cement; the second 2 ft., 5 of sand to 1 of cement; the next foot, 4 of sand to 1 of cement, and the last foot and also the mortar block (which stands up 12 in.) was mixed 3 of sand to 1 of cement. It was built on a good rock foundation, and the rock used in the concrete was firm and clean, the pieces being placed as close to each other as possible, while allowing the wet cement and sand to imbed each piece separately. The tamping of concrete is very important and also the mixing. It should be tamped until the water is drawn to the surface. In mixing a given amount of sand, generally measured by filling barrels and leveling off, it should be spread in a layer 4 or 5 inches thick on a mixing platform. The measured quantity of cement should be evenly distributed over this. The piles should then be turned over several times until thoroughly mixed; then it should be wetted down. The mortar-block should be left one inch low, and when the mortar is placed in position it is set 0.25 in. low and then grouted up with cement and sand mixed in the proportions of two to one. Then, when the concrete is sufficiently set the mortar is raised; a sheet of rubber, 0.25 in. thick, is placed on top of the mortar-block and all is finished. This gives a block which will last a lifetime. Sand for the concrete should be perfectly clean; and if it is not, it should be washed. A very small quantity of lime added to concrete in the mixing will be found to give it additional strength.

The stamps drop 100 times per minute, and have a capacity of between five and six tons per stamp. The mill-building is all steel and galvanized corrugated iron with plenty of windows. Power is furnished by a 120-horse-power Diesel oil engine. The mill is lighted throughout by electricity. All driving pulleys have friction clutches which permits of any one part being cut out without shutting down. The retorting is also done with oil for fuel. Owing to the high freight rates over the 60 miles of

desert, wood and coal were out of the question. In a quartz-mill no power is as satisfactory as steam.

The mill was contracted for by Harron, Rickard & McCone, of San Francisco. It was planned by M. P. Boss and constructed by the writer. The plant is running splendidly and saving a high percentage at a low cost.

## BATTERY FOUNDATIONS

By H. E. West

(June 2, 1904)

In the *Engineering and Mining Journal* of March 10 there is an article on a novel type of battery design under "Notes on a Cantilever Battery Frame," by Ira C. Boss. There are one or two remarks in connection with the building of the concrete foundation that engaged my attention. In the first place, it would appear from the description given that, although the efficient mixing of concrete is advocated, yet apparently the wet batter of cement and sand was added to the broken stone *in situ*. If this is a correct reading, it would not appear an advisable departure from the usual practice. Again, it does not appear to me that the "grouting in" of the mortar, after the concrete block has set, is an advisable method; this quite customary method in machinery-setting has, in the case of mortar-setting, been fruitful of trouble, as may be learned from the setting of mortars and anvil-blocks on concrete at the Alaska Treadwell mines and elsewhere. A more approved method is that given in a previous number of the *Engineering and Mining Journal*, where the writer advocated the substitution of a wooden block, an exact duplicate of the mortar, being accurately set on the yet pliable surface of the rammed concrete with an added surface made of a stiff mixture of sand and cement, the block being screwed down and leveled accurately in position and afterward removed.

With regard to that very prevalent practice of inserting a piece of $\frac{1}{4}$-in. sheet rubber, I cannot say that I at all see the necessity of this or any other material being inserted between the mortar and its block. It was all very well, in the days that I can well remember, when mortars were supplied rough from the casting. But now, when they are accurately surfaced, and the mortar-block is also a plane surface, it would appear to me that the very object of the concrete foundation — an absolutely

rigid base — is to some extent discounted by the resilience of the rubber sheet. As a matter of fact, I should imagine after some time that the actual "cushion" disappeared, which was the case with the older form of tarred blanket.

It would appear to me that an absolutely rigid connection between the plane surfaces of the mortar bottom and the block would be more correct in theory, and would yield better results in practice, than this present-day relic of "spring-beams" and "tarred blankets," the latter being necessary in the times referred to, when uneven surfaces were presented to each other. It would be interesting to ascertain what results have attended the discarding of any filling or joining material, such as described.

Support is given in this article to another very popular delusion, namely, the mixing of a certain amount of lime with cement, or *vice versa.* This is usually advocated for purposes of economizing the cement, not because it is beneficial to the cement. Seeing that cements are so carefully compounded, and that "overliming" is carefully guarded against, if this very indefinite statement regarding the strengthening of the concrete by the addition of a small amount of lime can be substantiated by the results of actual tests, it would appear that the cement used must have been of an inferior order, and that its composition was defective; since, if it were possible to augment its strengthening properties in setting, in which the time factor is also important, one would imagine that this should have been the careful consideration of the chemist at the manufacturer's works, and not have been left to be discovered in actual construction.

## BATTERY FOUNDATIONS

By M. P. Boss

(June 30, 1904)

The communication from H. E. West in the *Engineering and Mining Journal* of June 2, under the head of "Battery Foundations," touches so close to myself that I feel prompted to write a few lines. Being the designer, in general and in detail, as well as the co-contractor for the Tonopah mill in question, my responsibility in the premises was paramount.

It is not so much my purpose to explain the details of this specific case — for they may have already been answered — as it is to touch upon the principles involved, not for the sake of contention, but that my ideas may be known and accepted for what they may be thought to be worth. However, Mr. West's inference that the "wet batter of cement and sand was added to the broken stone *in situ*" is an error. The concrete was rammed down and coarse pieces of rock were individually and separately inserted and rammed with it — a practice I have always followed, and my battery foundations have always been thoroughly successful, except that the first (which was laid nearly sixteen years ago) was faulty from lack of careful ramming.

Personally, I never use lime in battery foundations. At the mill in question it was used only in a trifling quantity — in theory, I think, to counteract some fancied acidity of the water. As to grouting under a mortar, if it is carefully done on fresh concrete, it will set just as solidly as any other way of putting it in; but again, personally, I am satisfied to lay it full up and carefully sweep and dress the surface. In the first mill I put nothing between the sole plate (which I used then for the only time) and the concrete. In the second mill I put nothing between the mortar and concrete with four of the batteries, but under two mortars I put rubber.

I now always put rubber under mortars for the reason that

the vibration of the mortar, however small, slightly disintegrates the surface of the concrete, especially if wet, and the gasket entirely prevents such action. As to its being a cushion to the blow, that is infinitesimal when added to the weight of the mortar is the compression of four bolts of $2\frac{1}{4}$ in. in diameter — which is my standard. Anvil blocks under mortars I have always contended were illogical, and my own experience has demonstrated them to be absolutely unnecessary. They are objectionable because they are put in the form of a pedestal; if they were flat, the only objection to them would be that of unnecessary cost.

It seems to be a popular idea, even among engineers, that the great strain on a battery foundation is downward from the blow of the stamp. Now a single blow from the heaviest stamp, when spread over an area of two thousand square inches (the space occupied by the bottom of the mortar), gives a very low square inch pressure, so that it is almost wholly vibration from repeated blows that gives to us foundation problems. A mortar set upon a pedestal, even if the pedestal be of solid iron, has little to resist its vibration laterally, and from the top heaviness is more trying upon the concrete surface than where the mortar sets directly upon the concrete.

It is gratifying that criticisms are now upon details, and not the general idea of concrete foundations, since in building the first battery foundation of that kind I had to defend my idea single-handed, for I knew no engineer, mill-builder or mill-man who endorsed solid foundations for stamps.

## STAMP TAPPETS

By M. P. Boss

(October 13, 1904)

While in general features the stamp-battery is essentially what it was a third of a century ago, it has been touched up and refined in detail, and its efficiency and endurance greatly augmented thereby. What seem trivial details to a layman in mechanics aggregate in a machine so as to affect very materially its profit-earning qualities. Any one looking at so simple a thing as a stamp-battery of to-day can hardly realize that it has been the focus of many thinkers for many years. Nevertheless, the standard tappet as we use it possesses no detail that was not common to it many years ago. In fact, one detail commonly applied on the Comstock upward of thirty years ago, and fully appreciated at that time, is less generally used to-day, because manufacturers can save a little expense when it is not exacted. I refer to the counterbore.

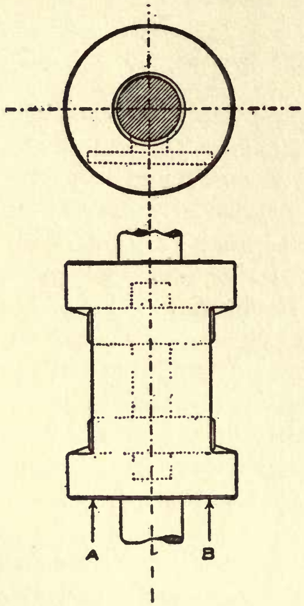
Of course, before the days of steel tappets much care was required not to split the tappet, and that was an added incentive to its employment, yet none will gainsay that heavy driving of tappet keys involves time and wear and tear. With a proper counterbore but a fraction of key pressure is required that otherwise is necessary to hold a tappet from slipping. I use the term "proper counterbore" advisedly, for, while most understand the "three bearing points" idea, few fully realize the wedge principle involved.

To demonstrate fully the two points referred to I add the drawings given in Figs. 8 to 11.

In Fig. 7 the line between *A* and *B* is at right angles to the center line intersecting the gib, and in the course of rotation of the stamp these points are alternately struck by the cam in operation. The blow being struck at a point outside of the

center line, through the pressure strain, clearly has a tendency to swerve the tappet from alinement with the stem. If the tappet yields in the least infinitesimal degree to this tendency, it more than returns when the blow follows upon the opposite side, and the tappet will work upward on the stem at each recurring change in alinement.

Fig. 8 represents the plan of a tappet without counterbore. As, necessarily, a tappet must be bored to an easy fit upon the stem, when key-pressure is but lightly applied against the gib, the contact of tappet-bore and the stem is but a narrow line, that gives weak resistance to the disalining influences referred to. Not until the key-pressure against the gib is made great enough to develop the elasticity of the metal in the tappet sufficiently to enwrap the stem with a broad line of contact opposite to the gib can the tappet resist the disalining influences of the heavy blows of the cam. Even a narrow counterbore aids this materially, as it gives two parallel lines of contact opposite the gib, and less key-pressure is required. This is one feature of the counterbore.



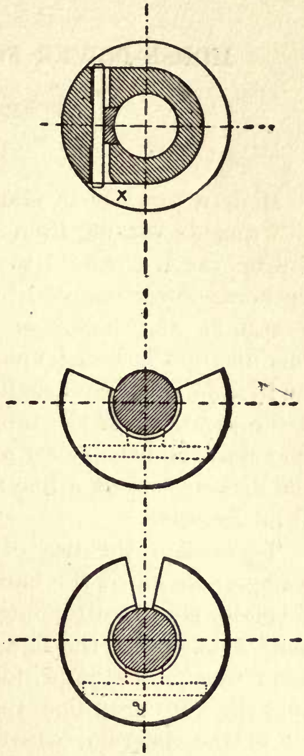
FIGS. 8 and 7.

Figs. 9 and 10 are the plans of tappets with counterbore. For the sake of clearness in showing the lines of contact to stem, the metal is shown cut away opposite the counterbore, thus emphasizing the contact points or lines.

Fig. 9 represents a narrow counterbore, and Fig. 10 a wide one. To think of the stem acting as a wedge between the two points of contact when key-pressure is applied is impossible with No. 9, but conceivable with No. 10. Lines tangent to the stem at points of contact correspond to the plane surfaces of the wedge it represents. So, if a counterbore extends 120 deg. — one-third of the circumference — every pound of pressure applied by the key is repeated at each of the two other points, whereas where there

is no counterbore, the key-pressure is repeated only at the one point opposite the key. Thus it is shown that a counterbore extending 120 deg. augments the pressure upon the stem 50 per cent. This is aside from the advantage of the widely apart parallel lines of contact-resisting disalinement.

Extending the counterbore beyond 120 deg., the wedging advantage gains rapidly, and it is not extravagant to say that a tappet on a 1200-lb. stamp could be successfully held by a single hard-wood key, by means of a counterbore of say 170 deg. of circumference. Not by any means is it to be inferred that this extreme is desirable, but a counterbore of 120 deg. or 140 deg. is economy in the long run.




FIGS. 11, 10, and 9.

Another feature in a tappet is to make the key-way openings easily distinguishable, the wide from the narrow. Rounding one side, as shown in Fig. 11, is simple and effective. Still another minor feature is that the inner sides of the flanges on a tappet should be more or less flattened, so that they may be struck by a heavy hammer, when moving a tappet, without battering or disfiguring its flanges.

## HORSE-POWER FOR TEN-STAMP BATTERY [1]

BY FRANK E. SHEPARD

(April 21, 1904)

Modern practice in stamp mills includes the use of stamps with weights varying from 500 to 1200 lb. per stamp, and hight of drop varying from 4 to 10 in. The diagram herewith shows the horse-power required for each 10-stamp battery for weights of stamps and hights of drop between the limits mentioned. This diagram is based upon 90 drops per minute, and includes the friction of the cam-shaft in the cam-shaft bearing, the friction of the cam against the tappets, the friction of the stem against the guides, and the power required for raising the stem vertically. The diagram shows a line for each hight of drop from 4 in. to 10 in. inclusive.

To explain the use of this diagram, I give the following example: Required the horse-power to drive a 10-stamp battery of 900-lb. stamps, dropping 90 drops per minute, hight of drop being 7 in. Find the figure 900 at the bottom of the diagram under weight of stamps, follow this line vertically until it intersects the 7-in. drop line, then proceed in a horizontal line to the left of the diagram, where it will be found that the required power for each 10-stamp battery is about 17.4 horse-power.

Another use of the diagram is in finding the change in horse-power by varying the hight of drop without changing the weight of the stamp. For example, with a 900-lb. stamp, in changing the drop from 7 to 8 in., the increase in power for each 10-stamp battery is from 17.4 to 19.4 horse-power.

The above powers are determined on the assumption that the bearings are properly lubricated and the stems in good alinement.

[1] Abstracted from the *Bulletin* of the School of Mines, Colorado; January, 1904.

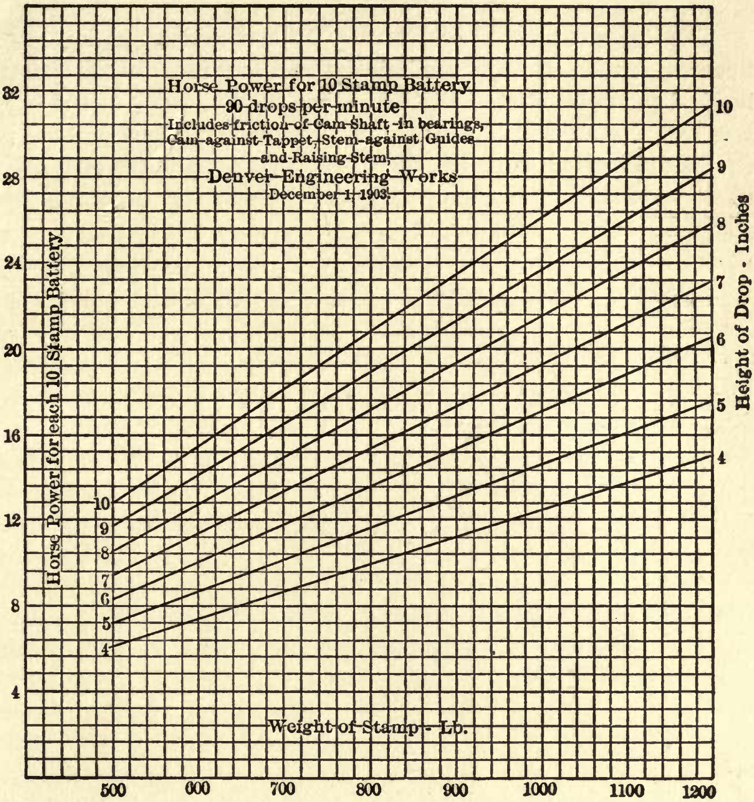

Horse Power for 10 Stamp Battery
90 drops per minute
Includes friction of Cam Shaft in bearings,
Cam against Tappet, Stem against Guides
and Raising Stem.
Denver Engineering Works
December 1, 1903.
Horse Power for each 10 Stamp Battery
Height of Drop - Inches
Weight of Stamp - Lb.
32
28
24
20
16
12
8
4
500
600
700
800
900
1000
1100
1200
10
9
8
7
6
5
4

FIG. 12.

## THE DRY CRUSHING OF ORE

(July 5, 1902)

The dry crushing of ore, usually to a considerable degree of fineness, is the necessary preliminary to a large number of metallurgical processes; but notwithstanding the antiquity of the subject and the extent to which it is practised, millions of tons of ore being crushed annually, there has been apparently but little effort to deduce the engineering data which are necessary for guidance in the design and operation of ore-crushing plants, although the scientific principles have been investigated and discussed by various writers. For this reason the paper on "Sampling and Dry Crushing in Colorado," by Philip Argall, read before the Institution of Mining and Metallurgy, at London, Feb. 20, 1902, is an especially important contribution to metallurgical literature.[1] Its value is enhanced by the facts that Mr. Argall is a recognized authority on this subject and the data now presented by him are deduced largely from his experience at the works of the Metallic Extraction Company at Cyanide, Colo., a plant designed and operated very successfully for seven years by him, where the fine crushing of the hard Cripple Creek ore was practised on a large scale. His keenness in investigation, clearness of perception and originality in overcoming difficulties have led to results which another might have failed to obtain. In presenting the following abstract of Mr. Argall's paper, advantage will be taken of the opportunity to refer to an also important paper entitled "Notes on Dry Crushing," by N. F. White (published in the *Transactions* of the Australian Institute of Mining Engineers, Vol. VI, pages 37 to 62) wherein valuable data of the results obtained at Mt. Morgan, Queensland, are described. Mr. White's paper, although much less comprehensive and original than Mr. Argall's, treats of the subject in the same practical way.

[1] For this paper, Mr. Argall was awarded, by the Institution of Mining and Metallurgy, the "Consolidated Gold Fields of South Africa, Ltd." Gold Medal and premium of forty guineas.—EDITOR.

Mr. Argall considers that in crushing material of which the pieces are smaller than 2-in. cubes, rolls are preferable to breakers. It is false economy and bad practice to attempt a great reduction in size in one operation or by means of one machine. His own rule is never to exceed the ratio of 4 : 1; that is, 16-in. pieces may be broken to 4 in., and the latter to 1 in. in another operation, but he would preferably provide three machines; for example, reducing from 16 in. to 5 in. by the first breaker, from 5 in. to 1.5 in. by the second, and from 1.5 in. to 0.5 in. by a set of rolls. The capacity of the various machines must of course be considered, so that under average working conditions each will be kept fully supplied with ore. The coarse breaker may be followed by two fine breakers, and these by one or more roughing rolls, while in small plants it may be more convenient to sacrifice efficiency by making the breaker reduce more than 4 : 1, to avoid the complication of a second machine for a small capacity. Mr. Argall does not go into the details of coarse crushing. He states, however, that a 12- by 20-in. breaker, reducing to a maximum size of 1.7 in., will easily have a capacity for 25 tons per hour of ordinary quartzose ores, but in case the ore is mostly in large pieces, say 12-in. cubes or larger, two breakers should be used in series with a screen between. Talcose and very wet ores give trouble; in bad cases they should be dried before going to the breakers. In referring to the capacity of crushing machines, it should be remarked how important a consideration the specific gravity of the ore is. The Cripple Creek ore, for example, is andesite, phonolite, and granite, occupying in lump form about 25 cu. ft. per ton and averaging 23 cu. ft. after reduction to 30-mesh size. In crushing such an ore, as compared with many sulphides, the mineral is not only harder, but its volume is much greater.

Mr. Argall discusses at considerable length the methods of ore sampling, which are naturally performed in connection with the crushing operation; in fact, a good deal of ore is crushed with that object alone in view. In this branch of his subject we shall attempt only to summarize his conclusions. He condemns unreservedly the antiquated method of quartering and ably points out its likelihood of introducing errors. If the work must be done by hand, the method of fractional selection — that is, the reservation of shovelfuls at regular intervals when the pile of ore

is being handled for other purposes — is not only more convenient, but also is far more accurate. Automatic machine sampling is, however, superior in all respects to hand-sampling. This is now the general consensus of opinion. In sampling ores containing 10 to 15 oz. gold per ton, Mr. Argall has found the following ratio between the average size of the ore cubes and the proportional weight of the sample to give accurate results:

| | | | | |
|---|---|---|---|---|
| Average size, inch | 1.0000 | 0.2500 | 0.0625 | 0.0171 |
| Proportion of sample % | 20 | 1.25 | 0.0785 | 0.0050 |
| Pounds from 100 tons | 40,000 | 2500 | 157 | 10 |

In practical work, however, larger quantities of the fine material would be taken, simply as an extra precaution. The following system of sampling proved to be quite successful: The ore leaving the breakers was of about 1 in. average size. From 100 tons (200,000 lb.) 40,000 lb. were cut out as the first sample. This was crushed to 0.25 in. size and 4000 lb. was cut out by the second sampler. This was reduced to 8-mesh size (0.0625 in.) and cut down by riffling to 250 lb., which was dried and crushed to about 30 mesh (0.0171 in.) and then riffled down to 15 lb. The last sample was pulverized to 90- or 100-mesh and riffled down to 1 lb., which was ground on the bucking-plate to pass a 120-mesh (0.004 in.) sieve and divided into samples for assay. If the work were well done a half assay ton from any of the 120-mesh pulps would check within 0.02 oz. (40c.) per ton. When the final samples were passed only through a 100-mesh sieve there was often difficulty in obtaining duplicate assays that would check. For cutting out the large samples the excellent apparatus designed by H. A. Vezin, which is now in general use in Colorado, was employed. It will be observed that the cutting down of the small samples was done entirely by riffling.

Mr. Argall remarks the following as some of the important points to be remembered in machine sampling:

(1) Take out a sufficient quantity in the first cut to represent accurately a thorough sample at that size.

Where the ores are of low grade, or very uniform in composition, a small sample will suffice. With iron ore, for example, or fluxing ores for blast-furnace work, where it is important to keep them as coarse as possible, 10 per cent. of ore as coarse as 6.-in. cubes can be taken out by the Vezin sampler, if necessary, and,

reduced proportionally, will give an accurate sample. When the ores are to be used in stamps or roller mills, and reduced to an ultimate state of fine division, it is preferable to reduce them finer for the first cut, and as a matter of precaution take out say 20 per cent. for the first sample.

(2) Always crush and thoroughly mix the ore between each cut, unless it is already quite fine, and in this case the greatest possible care must be exercised in thoroughly mixing before making the second cut.

The very essence of ore sampling is never to cut or reduce the ore a second time without first crushing to a degree of greater fineness. A moment's reflection will show the necessity of this. We will assume a lot of ore crushed to cubes of 1 in. average size, and that 20 to 25 per cent. is necessary to give a correct sample at this size, the latter (25) per cent. being of course taken as a matter of precaution. Now it is obvious that if this sample is reduced 50 per cent. without re-crushing, it simply amounts to taking out $12\frac{1}{2}$ per cent. in the first cut, which with 1-in. cubes we have found to be 50 per cent. too small to give a correct sample. It follows, therefore, that if 50,000 lb. are taken as the first cut from a 100-ton lot of 1-in. average cubes, and these are then crushed to $\frac{1}{4}$-in. average, 3125 lb. will give as accurate a sample at $\frac{1}{4}$ in. as 50,000 at 1 in.; or if the ore is all crushed to $\frac{1}{4}$ in., 3125 lb. will do for first cut; and further, that on a reduction to $\frac{1}{16}$ in., 195 lb. bears the same ratio between size of cube and weight of sample as the 50,000 lb. did to 1-in. cubes, and hence will give a correct sample.

Mixing comes next in importance, more especially for spotted ores; for unless the sample is well mixed it will require a greater number of cuts to give accurate results; that is, the speed of the cutter must be greater or the number of scoops increased.

When the sample is crushed in rolls and elevated to the cutter, the mixing is found to be sufficient, provided there is a steady feed to the rolls so that a uniform stream passes the cutters without intermission or break. When the ore is very fine, two or more cuts can of course be taken before crushing finer, but it is not nearly so safe a method as that previously described — solely on account of the difficulty in retaining a homogeneous mixture, more particularly when the ore is very dry.

(3) Use riffles for reducing the size of samples after leaving

the last automatic sampler. Abandon all forms of "coning and quartering," mixing samples on floors, scraping and sweeping up samples, etc., and thus eliminate these sources of error and labor-wasting devices.

In the fine crushing of ore it is essential to dry it down to a content of no more than 1 per cent. water, and also if the ore be clayey to raise its temperature to about 250 deg. F. Cold ore of that class lies dead on the screens and has a tendency to choke them, even if it be quite dry; but when hot they screen as well as hard, gritty ores. Mr. Argall employs the well-known 4-tube revolving drier, invented by himself. This drier is in principle the same as the ordinary cylindrical drier, the products of combustion from the fireplace at the discharge end passing through the tubes, over the ore, but it has the advantages that the ore is divided into four thin streams and the hot gases are brought more closely in contact with the ore. The Argall 4-tube drier is made in two sizes, No. 1 having a capacity of 80 to 100 tons of ore per day, and No. 2, 150 to 200 tons; the 6-tube driers have 50 per cent. more capacity. The efficiency of these driers is shown in the following table:

| Name and Location of Mill | No. of Drier | % Water | | Tons Ore Dried per 24 Hours | Tons Coal Used | Remarks |
|---|---|---|---|---|---|---|
| | | Before | After | | | |
| Bessie, Telluride, Colo.... | 2 | 8.00 | 1.22 | 177 | 2.66 | *a* |
| Cyanide, Leadville, Colo.. | 1 | 10.00 | 1.00 | 70 | 1.00 | *b* |
| Metallic, Cyanide, Colo... | 2 | 4.00 | 1.00 | .... | .... | *c* |

(*a*) Coal, poor quality of slack, burned with Jones underfeed stoker; ore clayey.

(*b*) Coal of fairly good quality, hand-fired; ore talcose and clayey; cylinders inclined 0.75 in. per foot and driven at two revolutions per minute.

(*c*) Coal good, burned with American stokers; ore silicious.

The above data show an evaporation per pound of coal of 4.54, 6.3 and 9 lb. of water respectively. In making such comparisons, however, it is necessary to remember not only the difference in the grades of the coal, but also that clayey ores require a higher temperature and are more difficult to dry than sandy or porous ores.

The cost of drying ore is comparatively little. Assuming an evaporative effect of 7 lb. water per pound of coal, the cost of drying 100 tons of ore containing 6 per cent. water, leaving 1 per cent. in the product, is as follows: 1430 lb. coal @ $3.00 = $2.14; labor, $2.00; repairs and lubricants, $0.43; motive power, etc., $0.25; total, $4.82, or approximately 5c. per ton.

Mr. Argall calls attention, as we have done repeatedly, in this journal, to the inaccuracy of specifying the size of wire cloth by the number of meshes per linear inch, the diameter of the aperture, which depends upon the gage of the wire from which the cloth is woven, being the important thing. Ordinarily the heaviest standard wire to be had for any given mesh is employed in ore milling, but there are cases where the opposite is the better practice. As a general thing the heavier wire is the better for coarse, gritty ores, but for soft, clayey ores, likely to choke up the screens, the finer wire is preferable in a dry crushing mill. For coarse screening, say down to 0.25-in. openings, perforated steel plate trommels of circular section give the best service. From 0.25 to 0.10-in., wire cloth trommels, also of circular section, are preferable. For the finer meshes the hexagonal form of screen, of light construction so that the weights or hammers, when they fall, will throw the whole sheet into vibration, and thus tend to keep the meshes open, are most advantageous. Heavy, rigid screens are a mistake.

The proper peripheral speed of hexagonal fine screens is about 180 ft. per minute. The angle of slope should not exceed 10 deg. from the horizontal. Ample screening capacity should be provided, making proper allowances for the inferior efficiency of dry sifting screens as compared with wet work. In crushing to 30 mesh (0.0171 in.) Mr. Argall has found that in general work 1 sq. ft. of screen will deliver about 6 cu. ft. of product per 24 hours, but with clayey ore the ratio is likely to fall to 1: 5. Large screening surfaces not only means greater output, but also less repairs. In crushing 99,270 tons of ore, the cost of maintenance of screens was 2c. per ton. With respect to housing the screens the following suggestions are offered:

"The screening process is usually a dusty one, in fact the most dusty in the mill. If, however, the screens are grouped on each floor and completely housed in, not in boxes, but in a room large enough for men to enter and walk freely around the screens,

to change and clean them, there will be no trouble from dust escaping, particularly if the room is connected with an exhaust fan, as it should be. Such a screen room would be closed everywhere air-tight, and have at most but two doors, one on each side; it could be wired for electric light, to be used only when the screen men are in the room. The heads of the elevators should be connected with this room, or rooms, and in fact all points in the mill requiring exhaust ventilation. The discharge from the exhaust fan should be conveyed into a bag-house, and forced through cotton bags, leaving the ore particles inside. The bags should be shaken every second day to detach the dust, and thus prevent the pressure against which the fan delivers the dust-laden air from increasing above 2 ounces. A volume of 20,000 cubic feet of air per minute would require 5000 square yards of filtering fabric."

For the purpose of fine crushing Mr. Argall is unequivocally in favor of rolls, except in connection with amalgamation processes, for which he recommends them only as an intermediate machine between the breaker and the stamps. He considers that the latter application would be advantageous, inasmuch as reducing the 2-in. stuff from a breaker to 0.75-in. size by means of rolls, and feeding the product of the latter to the stamps, the capacity of the stamps will be, in many cases, increased about 25 per cent., while the wear and tear on them will be reduced. In crushing for the cyanide and chlorination processes, in which a granular product is desired, stamps would hardly receive consideration at all in modern practice. Aside from rolls, about the only standard type of machine available is the ball-mill, the merits of which Mr. Argall discusses later on. In ore-dressing works rolls can be used in wet crushing down to 20-mesh with very good results, and with fair results down to 40-mesh if there is not much clayey matter in the ore. In considering the questions pertaining to the design and operation of modern rolls, as to which Mr. Argall is a high authority, it is best to use his own language:

"Rolls are usually described as belted or geared. Geared rolls of high speed are an abomination, even in coarse crushing. Belt wheels of ample size to transmit the necessary power should in all cases be used, with the result of a saving in power, a great saving in repairs, better and more uniform work, and almost

entire absence of the noise and jar inseparable from the use of geared rolls.

"Rolls as usually built have many defects, which in ordinary work are for a time, while the machine is new, passed over; but in fine crushing they immediately give trouble, and soon become intolerable. The first and most serious of these defects is separate journal boxes for the sliding roll, each held in place with its own tension rod and spring. In this arrangement it is impossible to keep an even pressure on the rolls; as, for example, one side or pair of journals may be held up to the crushing position with a spring pressure of say 30 tons, while the opposite side may have but one-half the pressure, the result being unequal opening of the roll across its face, inferior crushing, end thrust and hot boxes. Any lack of parallelism between the rolls results in end thrust and tendency towards wearing away of the collars, and allowing the rolls to pass each other, setting up flanging, and greatly augmenting end thrust. As an alleged cure for this state of affairs, we sometimes find one roll two or three inches wider than the others, so that the narrower one can wear into its fellow, forming a flange on either side. This is a palliative nearly as bad as the disease, and is in no sense a cure. It does, however, increase the friction, decrease the capacity of the roll, and increase its repair bill.

"Rolls should not only open parallel across the face under all conditions of service, but should also remain truly level; that is, a plane passing through the center of the fixed roll shaft should always intersect the center of the swinging or moving roll shaft. Any departure from this plane also tends towards end thrust, flanging, and greatly increasing frictional resistance of the machine. When the moving roll is mounted on any pin-jointed lever arrangement, these exact conditions are not fulfilled, while a slight wear of a pin joint disturbs the horizontality of the axis and increases the friction. Therefore, it is apparent that the movable roll is best mounted in a sliding device, with large anti-friction surfaces. Where the roll is mounted on pin-jointed levers, I consider it bad practice to have them unequal except the shorter movement is given to the spring; in other words, the opening of the roll should not be multiplied on the spring, but, if possible, reduced, so as to ensure smooth running and lessening of shock. All rolls should have swivel or ball and socket journal boxes.

The application of more power to the fixed than to the movable roll is not based on any good reasoning. Attempts to run one roll faster than the other are objectionable for several reasons; while catalog cuts showing an 8-in. belt on a 3-ft. pulley running the movable roll 101 revolutions per minute, against the fixed roll with a 14-in. belt on a 7-ft. pulley at 100 revolutions, are perhaps not more absurd than some other attempted roll practices, yet it has always struck me as very humorous.

"There is a wide variation in practice as to the speed of rolls, ranging, as they do, from the 30 or 40 ft. per minute of the old Cornish roll, to the 800 to 1000 feet of the modern high-speed rolls. In discussing speed, one very seldom hears of graduating it in accordance with the size of the ore to be acted on; yet this is, in my opinion, a fundamental principle. A careful series of experiments has convinced me that there is a speed for each size of material which gives the maximum capacity with the minimum power. These speeds were correlated, and from them the formula and diagrams presented herewith were deduced. I do not wish to be understood as saying that these are the only correct speeds at which rolls should be operated; but I do say they are the speeds I have found to give the best results; that they are safe and reliable, and the engineer who conforms his practice accordingly will not be disappointed.

"In reducing coarse ore with rolls, I never exceed a ratio of over 4 : 1, crushing, say from 2 in. to ½ in., ½ in. to ⅛ in., and so on. In the diagram (Plate 13) I assume a roll is set with ½-in. opening between tires, and crushing from 2 in. to ½ in., and so on proportionally for the other rolls. In each case the space between the rolls is equal to the mesh to which the roll is crushing. There is also, or rather there should be, a relative proportion between the diameter of the rolls and the size of the particles fed to them, more particularly in crushing the larger cubes, as it is manifest that if the size of the ore cubes materially exceed the 'angle of nip,' they will merely dance around and will not be drawn down and crushed. This phenomenon is more pronounced in the high-speed rolls.

"In my diagram the speed curves of the various size rolls are terminated on the right by another curve, which we may call the curve of nip. Taking the speed curve for a 42-in. roll, we find it terminated by the curve of nip on the 2-in. cube line at the

twenty-eighth ordinate, showing that to crush 2-in. cubes we require a 42-in. roll, and that its proper speed is 28 revolutions per minute. Taking a 26-in. roll, we find the maximum sized feed 1.25 in., the speed for these cubes 55 revolutions per minute, while for ½-in. cubes the speed is 73 revolutions, for 0.25-in. cubes, 88, and for 0.05 in., 108 revolutions per minute.

"The theoretical capacity of rolls might be described as the number of cubic feet per hour that would be rolled out in a ribbon, the length being the peripheral travel of the roll in one hour, the width that of the roller faces, the thickness being the distance

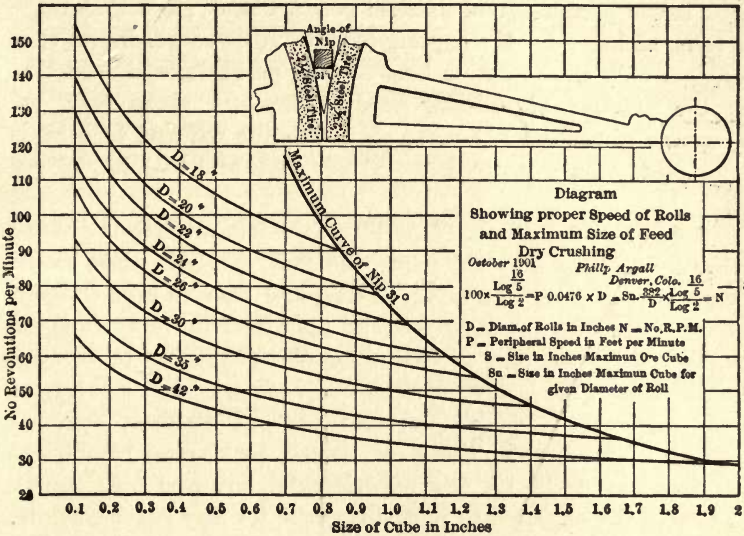



FIG. 13.

between the rolls; this we may express mathematically as $[P \times W \times S \times 60] \div 1728 = C$, wherein P represents the peripheral speed in inches per minute, W width of the roll face in inches, S space between rolls in inches, C capacity in cubic feet per hour. Such a ribbon could not of course be homogeneous; it would have spaces and cavities unfilled, and would consist of particles of every size from the largest that could pass through the space between the rolls down to the finest dust.

"Let us take the case of a 26- by 15-in. roll, 60 revolutions per minute, crushing from 1 in. to ¼ in. The theoretical capacity

is by the formula at once found to be 589 cu. ft. per hour, taking the mean diameter of the roll at 24 in. But how shall we find the actual capacity of these rolls in cubic feet per hour of *finished* product? It is perhaps best to be frank and state that we cannot, as there are too many variables to be taken into consideration; but we can closely approximate it.

"Following up the case I have taken, the maximum size cubes are 1 in., the minimum just a little coarser than $\frac{1}{4}$-in. as fed to the roll. Now, as different varieties of ores do not break alike, one sort may have as much as 15 per cent. more of say $\frac{1}{2}$-in. cubes in the feed than another, and would consequently give a larger percentage of finished product after passing through the rolls, and so on. My experiments have, however, shown that there is a very close relation between the percentage of reduction and the amount of finished product for any given ore. By percentage of reduction I mean an inch cube reduced to $\frac{3}{4}$ in. is 25 per cent. reduction; to $\frac{1}{2}$ in., 50 per cent. and to $\frac{1}{4}$ in., 75 per cent.

"Referring to diagram, Plate 14, on the left, an inch is divided by horizontal lines into 100 parts, the scale extending two inches in hight; next there is a series of diagonal lines to give the percentage of reduction at the given sizes; and lastly a heavy diagonal line marked 'Percentage of finished product for given percentage of reduction.' This curve of *finished product* I have fixed from actual experiments with quartzose ores of medium crushing qualities. I consider it, therefore, correct for average conditions with first-class rolls. A few experiments, however (say three), will enable the engineer to plot this curve for any particular ore, and thereafter he can quite closely determine the actual capacity of any given roll on that particular ore.

"Applying this diagram to our specific case, 1 in. to $\frac{1}{4}$ in. Following the diagonal line from 1 in. on the left, we find it intersects the $\frac{1}{4}$-in. horizontal line at the ordinate marked 75 per cent. reduction, showing that the maximum reduction in the crushing process has been 75 per cent. Taking 75 per cent. reduction on the right hand of the diagram, and following the horizontal line, we find it intersects the curve of finished product at the 30 per cent. ordinate, showing that for 75 per cent. reduction the finished product per hour is 30 per cent. of the theoretical. The latter we have previously seen is 589 cu. ft. per hour, 30 per

cent. of which gives the cubic feet per hour of finished product as 176 cu. ft., and so on for any ratio of reduction shown of the diagram.

"Diagram Plate 15 shows the capacity in cubic feet per hour of various size rolls, running at the speeds most suitable for the size of the feed they are assumed to receive, compiled from Diagrams 13 and 14.

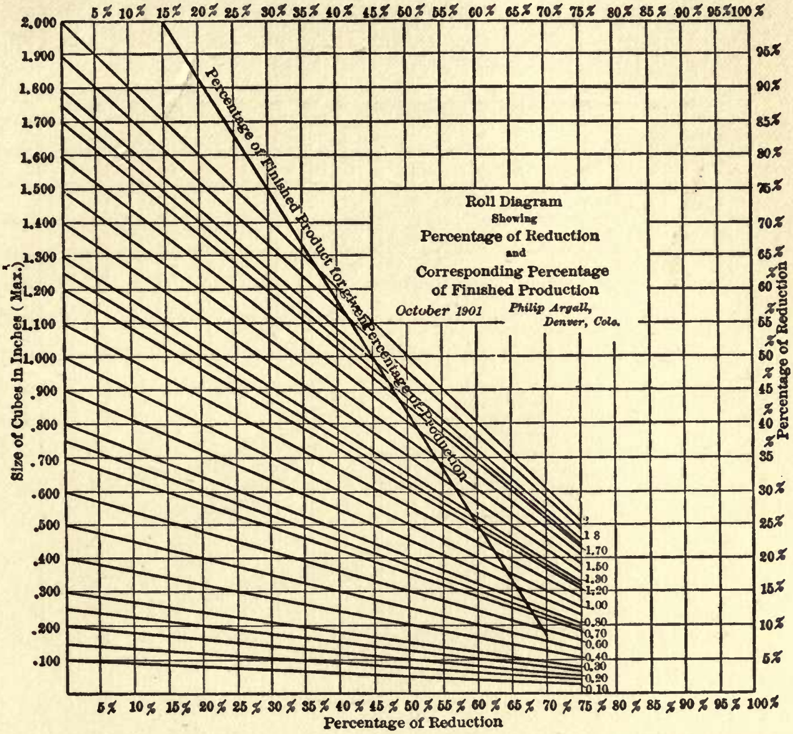



FIG. 14.

"These diagrams are figured in cubic feet per hour, which, in my opinion, is the only correct basis of comparison between ores. It is obvious that a roll crushing 5 tons of ore per hour, weighing 85 lb. per cubic feet in the crushed or finished state, would manifestly crush 10 tons per hour of material weighing 170 lb. per cubic feet, assuming the crushing condition of both ores is the same."

Mr. Argall gives some interesting data as to the excellent rolls designed by himself. He states that one set of these, which has been in hard service for nearly two years, day and night, has neither developed defects nor suggested improvements. For six months it was operated at a speed of 900 ft. per minute, crushing from 0.1 to 0.02 in., but was afterward reduced to a speed of 750 ft. to conform to the practice of that particular mill. All rolls, except coarse or roughing rolls, taking the ore from the breakers, should be provided with mechanical feeders. From 0.25 in. upwards, the stream should not exceed in thickness the

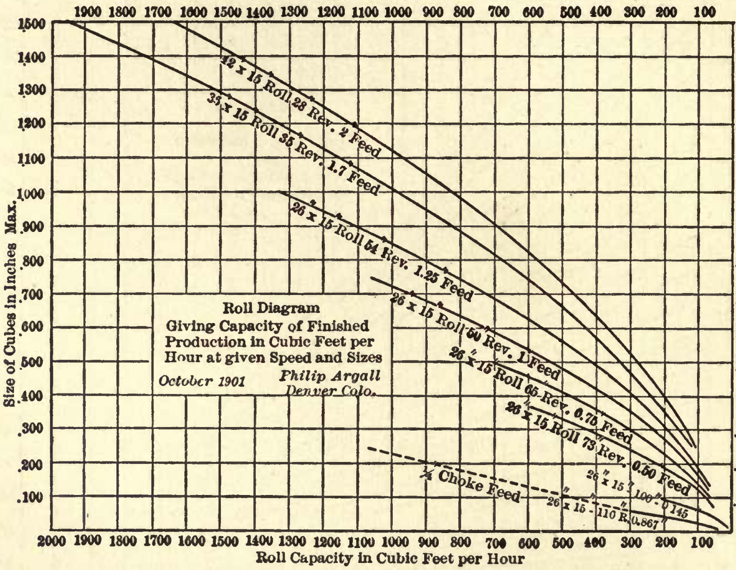



FIG. 15.

maximum faces of the cubes; below this size, however, thicker streams can be used, on the principle that Mr. Argall calls "choke feed," so that the ore particles are crushed upon each other in passing the point of contact, and the capacity of the rolls is very much increased. A 26- by 15-in. roll at 110 revolutions, crushing from 0.1 to 0.02 in., has a theoretical capacity of only 86 cu. ft. per hour, about 30 cu. ft. of finished product, whereas if run with choke feed 0.25 in. thick, its capacity will be 75 cu. ft. per hour to 0.02 in., using in each case first-class rolls. The space

between the boxes of the movable and fixed rolls on either side should be filled up solid with chuck plates of different thickness, and one wedge plate to give the fine adjustment. The rolls are next spaced to about the size of the finished cubes, say 0.02 in.; the tension rods are then screwed up to the crushing pressure desired. This pressure does not come on the journals, but on the chuck plates; the rolls can then be revolved without touching each other, but, immediately the 0.25 in. "choke feed" is turned on, they are forced apart against the accumulating spring pressure, and the ore is crushed upon itself, and also against the faces of the rolls. Mr. Argall used rolled steel tires 2½ in. thick, giving 2 in. of wear. They cost 8c. per pound, or rather averaged 8c. per pound during a period when 40,000 tons were crushed, the cost per ton for tires being \$0.0107 = 2.14 oz. per ton crushed; during the preceding half year 46,000 tons were crushed for a tire cost of \$0.0256 per ton; the average was therefore \$0.0181 per ton crushed, or 3.62 ounces.

Babbitt is better for roll bearings than bronze; it is also much cheaper. Where ball and socket, or swivel boxes are used, as they should be on all rolls, the boxes are preferably re-babbitted each time the tires are changed. It is important to have but one size of rolls, so that fewer stores are required, and one extra set of shafts and shells with swivel boxes can be used in any of the rolls. In this way shells can be changed, turned up, and boxes babbitted at leisure; and every time a change is made in the mill, the shafts have new babbitt bearings to run in. The adoption of this method obviates all trouble from hot boxes.

Mr. Argall emphasizes the importance of gradual comminution by means of a series of rolls, with screens interposed between each set, as a fundamental principle in the fine crushing ore, and criticises the so-called unit system described by John E. Rothwell in "The Mineral Industry," Volume IX, page 360, pointing out that it is more expensive in first cost, operation, and maintenance than a system designed for gradual comminution, and is not after all a complete unit, it being admitted of course that the unit system is a good thing if carried out in its entirety. However, the mill designed for gradual comminution is capable of such a development, and in fact in large installations is commonly so arranged.

One of the most interesting parts of Mr. Argall's paper is that

in which he outlines the design of a plant capable of crushing 400 tons per 24 hours of ordinary quartzose ore to pass a 26-mesh screen (No. 26 wire) and estimates the cost of its operation. Such a plant is best arranged in duplicate, 200 tons per day making a convenient unit, which will be capable of independent operation. For coarse crushing and sampling, a 12- by 20-in. breaker, reducing to 1.7-in. size, is provided, giving an easy capacity of 25 tons per hour with ordinary ore; in case the ore comes mostly in lumps of 12-in. or more, however, two breakers should be used in series with a screen between them. Where talcose and very wet ores are to be sampled in quantity, a drier should be provided, as it is often impossible to crush and sample such ores in their wet state, and in bad cases they should be dried before going to the breakers.

Following the breaker or breakers, a 36-in. by 16-in. roll, at 35 revolutions per minute, will give about 600 cu. ft. per hour through a 0.75-in. screen. For sampling purposes, the product of the 36-in. roll is passed over a Vezin sampler, and 25 per cent. taken out for the sample, say 120 cu. ft. per hour, which should be deflected to a 26- by 15-in. roll to be crushed to 3-mesh 10 wire, 0.1983 in., or, in round numbers, 0.20 in. At this size about 6 per cent. is ample for a correct sample, but, to be quite safe, one-tenth or 12 cu. ft. per hour may be cut out; if the ore does not contain over 5 per cent. moisture, the sample, 2½ per cent. of the original, can be passed directly to a fine grinder, reducing it to 10-mesh 18 wire, 0.0525 in., and again cut to one-tenth = 1.2 cu. ft. per hour, or 0.2 per cent. of the original volume; if damp, the sample is then dried and ground, then passed over a small Vezin sampler, or riffled, as found most desirable, being finally finished in the manner previously described.

The Vezin sampler with the necessary operating gear costs about $100. It requires a fall of about 6 ft. It can be installed, together with one set of rolls, a sample grinder, and a riffle sampler for $500 to $750.

Such a sampling works should easily handle 200 tons in 10 hours, provided the lots are of 50 to 100 tons each. Time is always lost in cleaning up between two separate lots of ore, and must be allowed for. Two such units will take care of the sampling of 400 tons per day of 10 hours with comparative ease. It is best to do all the sampling in the daytime. The methods of

weighing the ore and determining its moisture contents at the works of the Metallic Extraction Company were as follows: The ore was usually weighed on railroad track scales with self-registering beams; when the beam is balanced, a soft paper card is slipped into a slot and the lever pulled down, stamping the gross weight on the card. The car number, lot number, and date are then written on the card, which is filed away until the car returns unloaded, when the same operation is repeated for the tare. The weighmaster enters the weights in his book and sends the card to the settlement clerk, who keeps it until the lot is settled for. The card helps to settle many disputes as well as prevent errors in reading the beam. The moisture sample is taken from the rejected portion of the ore passing the cutters, say three times for each car, taking equal quantities and placing them in a can. When the last is taken, the can is shaken to mix the ore and 1 lb. is then weighed up in the presence of the seller, packed in a tray and put in a steam-heated drying closet and kept there, locked up, usually for 24 hours, when it is weighed again in the presence of the seller. In the case of talcose ores, which are difficult to put through the crushers without a preliminary drying, the moisture sample is best taken from the car immediately after weighing.

Each unit of the fine crushing plant will comprise one drier, three elevators, and four sets of 26- by 15-in. rolls, with the necessary screens, etc. The rolls are arranged in series, *a* reducing from 0.75 to 0.25 in., *b* from 0.25 to 0.1085 in., *c* and *d* from 0.0185 to 0.02 in. Roll *a*, at 65 r. p. m., will give a finished product of 222 cu. ft. per hour, of which 60 cu. ft. should pass a 0.1085-in. hole (5-mesh screen) and will, therefore, go directly to rolls *c* and *d*, leaving 162 cu. ft. per hour for roll *b*. Roll *b*, at 90 r. p. m., will give 160 cu. ft. per hour through a 5-mesh screen. Of the material reduced to 5-mesh size, 60 + 160 = 220 cu. ft. per hour, 75 cu. ft. will already be pulverized to 0.02-in. size, so that rolls *c* and *d* will have to take care of 220 − 75 = 145 cu. ft. per hour. These rolls at 110 r. p. m., with 0.25-in. choke feed, will each finish 75 cu. ft. per hour under ordinary working conditions. The capacity of the four sets of rolls, reducing from 0.75 to 0.02 in., will be, therefore, 220 cu. ft. per hour = 9.5 tons (reckoning 87 lb. per cu. ft.), which will correspond to 200 tons in about 21 hours, or at any rate easily in 22 hours. The power required

will be as follows: Coarse crushing and sampling, 35 indicated horse-power; fine crushing, 50; friction of engine and shafting, 20; total, 105 indicated horse-power. The coarse crushing and sampling mill require 70 indicated horse-power while running, and a power plant large enough to allow for that must be provided for each unit; this part of the plant will run only half time, however, which gives an average of 35 indicated horse-power per 24 hours. In crushing 200 tons of ore to 0.02-in. size in 24 hours, the work of 1 indicated horse-power is 400,000 ÷ (24 × 105) = 158.73 lb. per hour. In the fine crushing alone it is 400,000 ÷ (24 × 50) = 333 lb. per hour. These data are from actual results in practice. In comparing the efficiency of crushing plants, it is obviously necessary to reckon all the power required; not merely that for the crushing machines themselves, but also for their accessories, such as screens, elevators, etc. It will be observed that Mr. Argall's figures are based on the total power requirement. Mr. Argall is also precise in reducing his results to pounds per hour, which is a definite statement, whereas the expression of tons per day is not. In the latter case the reader is left in doubt as to whether the ton is of 2000, 2204.6 or 2240 lb., while the day may be anything — 8, 10, 12, 20, or 24 hours.

The cost of crushing 400 tons of ore per day to 0.02-in. size in such a two-unit plant will be as follows: Coarse crushing and sampling — labor, \$0.04325; waste and lubricants, \$0.00620; brooms and brushes, \$0.00060; tools, \$0.00210; sundries, \$0.00060; total operating expense, \$0.05275 per ton. The maintenance will come to \$0.05420, divided as follows: Labor, \$0.0232; fittings, \$0.0006; nails, bolts, and screws, \$0.0010; timber, \$0.0015; iron and steel, \$0.0017; belts and lacing, \$0.0041; castings (pulleys, gear, etc.), \$0.0002; brass and babbitt, \$0.0003; elevator buckets and bolts, \$0.0018; chain and sprockets, \$0.0014; conveyors, \$0.0022; roll shells, \$0.0020; crusher repairs, \$0.0114; clutches, \$0.0008; sundry items, \$0.002. The total cost of maintenance and operation, not including power and general expense, is, therefore, 10.695c. per ton. The labor in operation per shift of 10 hours is as follows: One-half time of foreman, \$2.50; 2 men at breakers, \$4; 2 men at samplers and conveyors, \$4; head sampler, \$3; assistant sampler, \$2; roustabout, \$1.80; total, \$17.30; \$17.30 ÷ 400 = \$0.04325. It is assumed that the ore is delivered to the breakers.

*Fine Crushing.* — The operating expense per ton of ore is as follows: Labor, \$0.07625; lubricants and waste, \$0.01510; tools, brushes, and brooms, \$0.01300; coal for drying, \$0.02140; total, \$0.12575. The cost of maintenance is as follows; Labor, \$0.0446; nails and screws, \$0.0005; lumber, \$0.0004; brick and fire-clay, \$0.0015; iron and steel, \$0.0118; belts and lacing, \$0.0243; pulleys and gears (castings), \$0.0102; babbitt and brasses, \$0.0623; screen cloth and perforated sheets, \$0.0206; elevator buckets and bolts, \$0.0088; chains and sprockets, \$0.0021; conveyors, \$0.0017; roll shells, \$0.0181; clutches and couplings, \$0.0006; sundry fittings, \$0.0025; total, \$0.15000. The grand total is therefore \$0.12575 for operation plus \$0.15000 for maintenance, or \$0.27575. The crew of the fine crushing department per shift of 8 hours is one man at the feeders and driers, \$2; one man at the 8 sets of rolls, \$2.50; one man oiling and sweeping, \$2; one man attending to screens, \$2; total per shift, \$8.50. Three shifts at \$8.50, together with the wages of foreman at \$5, come to \$30.50, and \$30.50 ÷ 400 = 7.625c. per ton.

The total cost is, therefore, summarized as follows: Coarse crushing and sampling, 10.695c. per ton; fine crushing, 27.575c; power (estimating \$72 per indicated horse-power per annum, the engine being non-condensing), 10.500c.; total, 48.77c., or, say 50c. in round numbers. This does not include general expense (administration, insurance, taxes, etc.), or any deduction for amortization.

It is interesting to compare the above figures with the data communicated by N. F. White as to the results at Mount Morgan, where fine crushing is done both by means of rolls and by ball-mills, the installation of the latter for this purpose being perhaps the most complete and extensive that has yet been made. The Mount Morgan ore, which is soft and friable, containing about 10 per cent. of hard quartz, is treated by the chlorination process. The first plant installed comprised a Blake breaker, a drier, and four sets of Krom rolls, together with the necessary screens, elevators, etc. The success of this plant led to the installation of another one, consisting of two units, each equipped with a Krom breaker and four sets of Krom rolls, arranged in series. The dimensions of the rolls and the distance of their faces apart were as follows: No. 1, 26 by 15 in., ⅜ in.; No. 2, 26 by 15 in., 3-16 in.; No. 3, 30 by 16 in., 1-16 in.; No. 4, 30 by 16 in., close

together. The ore was crushed to pass a 20-mesh brass wire screen, data not given, but with apertures probably of about 0.025 in. The cost of crushing per ton of ore was as follows:

| OPERATION | MILL NO. 1 | | MILL NO. 2 | |
|---|---|---|---|---|
| | S. | D. | S. | D. |
| Wages | 2 | 6.40 | 2 | 6.20 |
| Stores | | 2.80 | | 3.90 |
| Firewood | 1 | 2.50 | 2 | 0.50 |
| Cartage | | 2.30 | | ........ |
| Water | | 0.57 | | 0.52 |
| Electric light | | 1.43 | | 0.59 |
| General expense | | 1.18 | | 0.57 |
| Total | 4 | 5.18 | 5 | 0.28 |
| MAINTENANCE | | | | |
| Wages | | 3.52 | | 5.20 |
| Stores | | 10.67 | | 10.80 |
| Timber | | 0.22 | | 0.18 |
| Mechanics' work | | 6.28 | | 5.21 |
| Total | 1 | 8.69 | 1 | 9.39 |
| Grand total | 6 | 1.87 | 6 | 9.67 |

In the new mill the rolls were driven at 112 r. p. m. The wear of the tire steel was 0.108 lb. per ton of ore crushed. The capacity was 125 tons per day, and 100 indicated horse-power was required. (It is not stated in this paper whether the tons meant are of 2240 or 2000 lb., but presumably they are the former.) Mr. Argall computes that the work done at Mount Morgan, in crushing to 0.025-in. size, is 116.66 lb. per indicated horse-power hour, as compared with 243.6 lb. per indicated horse-power hour in the plant (of practically the same equipment as to number of machines), which he has outlined, crushing to 0.025 in., and presents these figures as a comparison between the latest Colorado practice with modern rolls and the Mount Morgan experience with less efficient machines. This assumes, of course, that the Mount Morgan ore is of about the same weight per cubic foot of finished product and of about the same crushing quality as the Colorado (Cripple Creek) ore. We think, however, that the data are lacking, not merely as to the character of the ore, but also as to the details of the Mount Morgan practice, to justify this deduction of more than double duty per indicated horse-power, inasmuch as the Krom rolls, although now greatly im-

proved upon by others, are, nevertheless, a high class of machine. It is obvious that Mr. Argall recognizes the deficiency in data as to the Mount Morgan practice and makes his computations rather for the purpose of illustrating the high efficiency of modern American roll-crushing than as an absolute comparison of types of rolls.

In a year's run with the 8-roll mill at Mount Morgan, during which 45,844 tons of ore were crushed, the average cost of operation and maintenance, not including electric light, water supply, breaking and drying, and presumably no general expense, was as follows: Screens — steel and brass wire-cloth, 2.62d.; flannel and calico, 0.02d.; belting and sundries, 0.86d.; total, 3.50d.; rolls — new tires, 2.01d.; repairs, 1.21d.; belting and sundries, 0.87d.; waste, 0.17d.; oil and kerosene, 0.46d.; tallow, 0.75d.; total, 5.47d.; elevators — buckets, belting, etc., 1.87d.; sundries, 0.27d.; total, 2.14d.; wages, 2s. 11.25d.; grand total, 3s. 10.36d. The labor per shift of 8 hours was as follows: Two men at breakers, 1 man at the two driers, 2 men at the 8 rolls, 1 man at screens, elevators, etc.; 1 overseer, 1 spare man, and one-third of the superintendent's time.

When the sulphide ore was opened at Mount Morgan, it was found that the wear and tear on the rolls increased excessively in crushing the much harder ore, while the output of the plant was reduced. This led to the installation of a plant of Krupp ball-mills, at first experimentally with four No. 4 machines. Although the ore was crushed in these to pass a 35-mesh screen, the cost per ton was very much less than in the roll plant. In a year's run the four mills put through 23,788 tons of ore, and the working time having been 313 days, the average per mill, per day, was 19 tons. Each mill required about 10 indicated horse-power. The success of this plant led to the installation of another and larger one, comprising 16 No. 5 mills, arranged in groups of four, each group having its own breaker, revolving drier, and the necessary elevators, etc. This plant is used for the treatment of the low-grade oxidized ore of the mine, the same which had previously been crushed in the roll plant.

The comparative cost of crushing oxidized ore at Mount Morgan with rolls and ball-mills, reducing the ore in each case to 0.025 in., based on one year's work, amounting to 45,844 tons for the rolls and 130,776 tons for the Krupp ball-mills, was as follows:

| RUNNING EXPENSES | 8 KROM ROLLS | | 16 BALL-MILLS NO. 5 | |
|---|---|---|---|---|
| | S. | D. | S. | D. |
| Wages | 2 | 6.20 | 1 | 0.034 |
| Stores | 0 | 3.90 | 0 | 2.766 |
| Firewood | 2 | 0.50 | 0 | 9.538 |
| Coal | ... | ... | 0 | 3.817 |
| General expenses | 0 | 0.57 | 0 | 0.223 |
| Electric light | 0 | 0.59 | 0 | 0.332 |
| Water supply | 0 | 0.52 | 0 | 0.068 |
| Cartage | ... | ... | 0 | 0.235 |
| Inclined tram | ... | ... | 0 | 1.317 |
| Total running expense | 5 | 0.28 | 2 | 6.330 |
| MAINTENANCE (REPAIRS) | | | | |
| Wages | 0 | 5.20 | 0 | 3.325 |
| Stores | 0 | 10.800 | 0 | 10.493 |
| Cartage | ... | ... | 0 | 0.038 |
| Mechanics' work | 0 | 5.210 | 0 | 2.906 |
| Timber | 0 | 0.180 | 0 | 0.015 |
| Total expense of maintenance | 1 | 9.39 | 1 | 4.777 |
| Grand totals | 6 | 9.67 | 3 | 11.107 |

The crew of the roll plant comprised 8½ men per shift, as previously stated. That of the ball-mill plant consisted of 4 men at the breakers, 2 men at the driers, 3 men at the mills, 1 man and 1 boy in the engine-room, and two men in the boiler-room, besides 1 man and a boy for general work, a total of 15 men per shift; but it will be observed that this includes the labor in the steam and power plant, which is not included in the statement as to the roll plant, though the cost of power is evidently included in the total of 6s. 9.67d. per ton. The No. 5 ball-mills in this plant crush an average of about 24 tons per 24 hours to pass a 20-mesh screen.

Previously in his paper, Mr. White stated that the plant of four No. 4 mills crushed 19 tons of sulphide ore per 24 hours to pass a 35-mesh screen. This relatively high performance is probably accountable to the greater specific gravity of the ore. The method of reckoning the product of crushing machines by the cubic feet, which is adopted by Mr. Argall, gives more accurate data and is never misleading. The character of the ore, of course, makes considerable difference in the duty of crushing machines. In crushing with rolls at Cyanide, Colo., the ores varied from hard jasper and chalcedony to soft andesites and porphyries, and included granite, phonolite, and quartz rock. Granite proved to be the most easily crushed, quartz next; the softer ores were less satisfactory. Rolls

crush exactly on the same principle as rock breakers, in which soft and tough ores give low capacity, while hard, brittle ores, that break with a snap, seldom give low capacity.

The power required by the plant of 16 mills was as follows: Sixteen ball-mills at 13 indicated horse-power, 208; 4 breakers 26; 4 driers and elevators at 6 indicated horse-power, 24; line shaft, countershaft, etc., 14; friction of engine at 18 per cent. of total of previous items, 49; total, 321. The capacity of the mill is 400 tons per 24 hours, and the work of 1 indicated horse-power is 116.29 lb. per hour as the maximum, but the average is 368 tons and the work of 1 indicated horse-power per hour 107.33 lb. The power is developed by means of a triple-expansion horizontal engine of 24-in. stroke and cylinders of 15, 24, and 39 in. diameter. This engine is directly connected with the line shaft. The steam is supplied by 5 Babcock & Wilcox boilers, of 120 horse-power each, the steam pressure at the boilers being 150 lb. and at the engine 145 lb. The consumption of coal is 2.58 lb. per indicated horse-power hour, which Mr. White considers very fair considering the quality of the coal and other unfavorable conditions.

The ball-mill is now generally conceded to be an efficient, fine pulverizer, and it has the advantage of combining in one apparatus, which is capable of receiving ore directly from the breakers and delivering it at the desired degree of fineness, the screening and elevating mechanism, which in a roll plant have to be provided independently. This must necessarily save considerable labor, besides affording some other advantages. On the other hand, the consumption of steel by wear of the various parts is undoubtedly very high in the ball-mills. The following table shows the wear of the principal parts of a No. 5 Krupp mill in pounds of steel per ton of ore as reported by Mr. White:

| | Total loss of steel | Loss in lbs. per ton | | |
|---|---|---|---|---|
| | | Plates | Belts | Balls |
| Hunch plates | 89,000 | .681 | .... | .... |
| D bolts | 4,035 | .... | .038 | .... |
| E bolts | 3,039 | .... | .023 | .... |
| Scoop or perforated grinding plates | 2,814 | .022 | .... | .... |
| G bolts | 321 | .... | .002 | .... |
| Cheek or side grinding plates | 18,900 | .144 | .... | .... |
| Square-headed bolts | 1,153 | .... | .009 | .... |
| Steel balls | 94,752 | .... | .... | .725 |
| Total wear of steel per ton crushed | 1.644 lb. | 0.847 | 0.072 | 0.725 |

The plates become useless when worn down about two-thirds of their original weight. This would reduce the actual wear of metal in the mill from 0.847 to 0.565 lb. per ton of ore, leaving an apparent actual consumption of steel carried off in the ore of 1.362 lb. per ton of ore crushed.

The following table shows the renewal of parts of 16 No. 5 ball-mills in twelve months, during which time 130,776 long tons of ore were crushed to pass 20-mesh screens, with the cost per ton for renewals, together with the approximate life of some of the parts:

| | Number used in one No. 5 mill | Number used in twelve months | Unit | Price s. d. | Cost per ton, pence | Approximate life, months |
|---|---|---|---|---|---|---|
| Steel balls | 112 | 5264 | each | 5 0 | 2.412 | ........ |
| Hunch plates | 40 | 1000 | " | 45 0 | 4.130 | 7½ |
| Cheek plates | 20 | 270 | " | 42 4 | 1.066 | 14 |
| Scoop plates, perforated and grinding | 10 | 80 | " | 80 0 | 0.587 | 18 |
| Bolts D | 40 | 1153 | " | 1 3 | 0.132 | ........ |
| Bolts E | 40 | 1013 | " | 1 3 | 0.115 | ........ |
| Bolts G | 40 | 214 | " | 1 3 | 0.024 | ........ |
| Bolts, square-headed | 60 | 769 | " | 1 3 | 0.088 | ........ |
| Fore sieve plates, set of five | 5 | 196 | set | 31 6 | 0.565 | 5 |
| Brass wire gauze (set of 10) sq. ft | 61 | 7876½ | ...... | 0 7½ | 0.451 | 1¾ |
| Flannel, lineal yds | ........ | 644 | ...... | 1 1 | 0.062 | ........ |
| Bag leather (sides) | ........ | 19 | ...... | 16 0 | 0.037 | ........ |
| Clout nails (packets) | ........ | 183 | ...... | 0 5 | 0.007 | ........ |
| Glass lubricators | ........ | 120 | each | 0 7 | 0.008 | ........ |
| Set screws | ........ | 227 | lb. | 0 5 | 0.008 | ........ |
| Bolts, 1½ in. x ⅜ in. | ........ | 269 | " | 0 3 | 0.006 | ........ |
| Bolts, 3 in. x ⅜ in. | ........ | 258 | " | 0 3 | 0.006 | ........ |
| Bolts, 3 in. x ½ in. | ........ | 295 | " | 0 3 | 0.007 | ........ |
| Washers | ........ | 752 | " | 0 2 | 0.012 | ........ |
| Screen frames, pine | 10 | 108 | ...... | ........ | 0.055 | 18 |

A comparison of the results at Mount Morgan show operating expense per ton of ore of 30.330d., in the case of the ball-mills versus 60.28 in the case of rolls; maintenance expense of 16.777d. versus 21.39d., a total of 47.107d. versus 81.67d. It appears, therefore, that although the consumption of steel in the ball-mills was 1.362 lb. per ton of ore crushed, against only 0.108 lb. of roll shells, the cost of maintaining the ball-mill plant was on the whole less than that of the roll plant. With respect to the operating expense, it should be remarked that 11d. occurs in the

item of fuel, which is chiefly due to the more economical power plant, while the boilers of the roll plant supply some steam for outside purposes. The fact that the quantity of ore crushed per indicated horse-power per hour is substantially the same in both mills shows that the ball-mills are not really entitled to the saving in cost of power which the Mount Morgan figures show. The advantage is chiefly in the item of labor in running expense, which is reduced 18d., or more than one-half. Mr. Argall calls attention to two other points, namely, that the roll plant was old, while its capacity was only about one-third that of the ball-mill plant. The former criticism is good; the latter not so good, inasmuch as the roll plant was substantially the same as what he outlines as a unit of maximum efficiency, wherefore the reduction in cost per ton by further increase in capacity would probably be not very important. The wages paid at Mount Morgan were not stated by Mr. White, saving the single remark that the men attending the ball-mills received 8s. per 8-hour shift, whence we may infer the rates are about the same as in Colorado. It appears, however, that a plant consisting of 2 breakers, 2 driers, and 8 rolls crushes only 125 tons at Mount Morgan, against 400 tons in Colorado. For the coarse crushing, the Mount Morgan plant has 2 men at the breakers per shift of 8 hours, while the Colorado plant has 2 men for a single shift of 10 hours. Each plant has 1 man per 8-hour shift at the driers, 1 man at the screens, and 2 men at the rolls (in Colorado, 1 man "oiling and sweeping"). The elaborateness with which the sampling was done at the Colorado plant makes precise comparisons impossible, but it is clear that the practical results at Mount Morgan are inferior. To what extent this is due to difference in the character of the ore, to the lower efficiency of the crushing machinery, to the less perfect general design of the works, to the less economical systemization of the labor, etc., conclusions can hardly be drawn from the present data. In the case of the comparative results between the rolls and ball-mills at Mount Morgan, however, the test is valuable inasmuch as the ore treated was identical, although all the conditions were not perhaps the same. Mr. Argall appreciates the advantages which ball-mills may have to those requiring small units, but for large capacities, say from 60 tons per day upward, he considers that rolls are vastly superior.

It is hoped that other engineers who are engaged in crushing

ore on a large scale will compile their results under different conditions with the same minuteness and careful analysis that Mr. Argall has done, and thus throw further light upon this important subject. To him is due, however, the thanks of the profession for presenting scientific data whereby the capacity of rolls for crushing any material to any degree of fineness can be computed with some degree of accuracy, and not have to be left to guesswork or the rule of thumb deductions from imperfect experience.

# MODERN CRUSHING AND GRINDING MACHINERY

By Philip Argall

(May 11, 1904)

The recent discussions on fine crushing and grinding with stamps, pans, and tube-mills appear to accentuate the principle of crushing by "successive comminution," a process that, with others, I have for many years not only advocated, but also put into successful operation.

The idea that all the stages of the process can be carried out by one machine, be it roll, stamp, or patent pulverizer — and their name is legion — appeals strongly, both to the uninitiated and the so-called practical man, "he who practices the errors of his predecessors," on account of its apparent simplicity. One machine is to do everything, that is to say, take rocks of 2-in. cube and crush them to pass a screen aperture of say 1-50 in.; if it amalgamates at the same time, so much the better; and should it be able to perform other functions, why that simply enhances its value, although adding greatly to its complications. The United States, with more inventors to the square mile than any other country, can lick creation on patent pulverizers and crushers that are able, unaided and alone, to crush and grind coarse rock to infinite fineness; and the list is being momentarily enlarged by well-meaning farmers, backwoods lumbermen, and ingenious mechanics, most of whom have never seen a mine, and who often lack even the elementary knowledge of the practical requirements in ore reduction. To these inventors it is well to state clearly that the single machine for fine-grinding coarse rock is fast passing into the friendly oblivion such machines are so eminently fitted to adorn.

Those of us who have investigated the mechanical efficiency of a given machine, meaning thereby the pounds of ore crushed per horse-power hour from, and to, stated maximum sizes, know that each machine of value has its economic limit, at which it

will give the maximum output in pounds per horse-power hour, and also that there are fixed ratios of reduction that should not be exceeded. These limits, however, are not in every case determined, yet I have no hesitation in stating that the machine which reduces a 2-in. cube to pass a 1-50-in. screen-aperture, in one operation, is a very inefficient apparatus, even if it should prove to be the most modern 1350-lb. stamp.

I have elsewhere shown that the efficiency of a rock-breaker falls off rapidly when a reduction exceeding 4 to 1 is attempted, say 8-in. to 2-in. cubes, and the same rule holds good for rolls; and that for reducing ores below two inches, rolls are vastly superior to rock-breakers, for the reason that, while both machines work practically on the same principle, one is an unbalanced, or, at best, imperfectly balanced, intermittent-action, reciprocating machine, and the other a perfectly balanced, continuous-action rotary machine. Large rolls could be constructed that would compete favorably with rock-breakers on sizes above two inches, but they would be heavy, cumbersome, and difficult to transport to the metal-mining regions, so I hold that the combination of breakers and rolls I have indicated best answers our present requirements in metal mines, where the rock going to the breakers is never of unusual size — as quarry rock, for example.[1]

The reason that a 4 to 1 reduction should not be exceeded in rock-breakers is obvious to any one, and to a great extent the same rule applies to rolls. The economic limit of rolls, crushing dry, is from 2 in. to about 0.02-in.; below this their efficiency falls off. With average quartzose ores I have reduced with four rolls in series 330 lb. per horse-power hour from 0.75 to 0.02-in., crushing dry, and I believe practically as high an efficiency can be attained in wet crushing with a properly designed roll.

Now, it may be asked, what is the economic range of stamps? Some say, it matters not whether the ore is fed in pieces up to cubes of 2-in. maximum, or in cubes as small as ½-in., the capacity through a given screen-aperture will, in either case, be the same; while others claim a larger output for the ½-in. feed; good men can be found supporting either side of the argument. Now, I

[1] The Edison giant rolls are employed in some large mills, among them the mill of the New Jersey Zinc Company at Franklin Furnace, N. J., instead of rock-breakers. — EDITOR.

hold that recent practice has quite wrongly placed on the stamp much of the work that can be better performed by breakers and rolls, and this error has been intensified by constantly increasing the weight of the stamp, so that it can smash any size of rock that enters the mortar. Passing to the other extreme, the stamp is not an efficient fine-crushing machine — never was, nor from its construction can it ever be. It is just possible that a 1250-lb. stamp might sink through a ½-in. bed and even strike the die, and so a well-prepared ore might, under these conditions, show even a lower stamp-efficiency than if the batteries were fed with a maximum cube of two inches, as in the latter case there would be material on the die that would resist the weight of the falling stamp, to the end that the greater part of the energy would be expended in the useful work of crushing ore. Supposing, however, that ½-in. cubes are fed to, let us say, an 800-lb. stamp, and the output from the latter stamp was fully as great as that of the heavier one (as in most cases I know it would be), then the stamp advocates would be compelled to admit there was something wrong. So it becomes necessary to reason out whether stamps or rolls are the more efficient machine for reducing ores from 2-in. to ½-in. cubes, and to this there can be but one answer, as all must admit the stamp is not a coarse crusher; we have previously seen that it is not a fine crusher. Hence, if it has an economic range comparable with modern machines, it must be somewhere between ½ in. and 1-50 in., and even within this range I maintain rolls are more efficient, even if the stamps are used in series in order to secure in their operation the benefits of "successive comminution."

The trouble with stamps — apart from their being reciprocating machines in which a dead weight of 1000 to 1350 lb. has to be picked up from a state of rest by a rapidly rotating cam 100 times per minute — is that modern practice attempts to do too much with them; crushing ore from 2 to 0.02 in. is a reduction of 100 to 1, against say 4 to 1, for other classes of crushing machines. Surely this enormous range of reduction cannot be sound practice. Therefore, I hold, the modern practice of increasing the weight and range of reduction of the stamp is all *in the wrong direction*, and that in attempting to do the work of rock-breakers or rolls with a stamp, no matter what its weight, the results will invariably be a much more expensive plant, and one of greatly

impaired efficiency, as measured by the weight of rock crushed per horse-power hour.

To lift 1250 lb. to crush a 2-in. cube of hard rock may be defended, but to use the same weight on, say, 0.08 and 0.04-in. particles, to reduce them to pass a 1-50-in screen-aperture, does not appeal to any sense of fitness. This brings up another point: large pieces of rock yield best to a crushing force, smaller ones to grinding, and while there is some grinding action between the large and fine rock in a stamp-mortar, it is on the whole insignificant, and for this reason pans and tube-mills are vastly more efficient for reducing sands and fine ore than stamps or any other form of crushing or impact machines.

I do not desire to establish any set figures regarding the weight of stamps or the fineness of the ore fed to them; such factors will depend somewhat on the nature of the ores treated in any given case, and can be worked out by those in charge of stamp-mills.

The accompanying diagram shows the efficiency in pounds per horse-power hour of stamps and rolls crushing from about 2 in. to 1-50 in. maximum. I would point out, however, that the stamp will give a greater proportion of fine material; in other words, in discharging through the same size screen-aperture the stamp will crush from 10 to 15 per cent. finer than the roll — used as a wet crusher — and this must be allowed for, as useful work done.

My idea of a roll plant to reduce ores, dry, from 2 in. to 1-50 in. would be as follows:

| | | | | | |
|---|---|---|---|---|---|
| Rough rolls | from | 2 | to | 0.75 | in. |
| Second " | " | 0.75 | " | 0.24 | " |
| Third " | " | 0.24 | " | 0.08 | " |
| Fourth " | " | 0.08 | " | 0.02 | " |

Taking a crushing and amalgamating plant in which the ore is finally reduced to 1-400 in. for filter-press work, I would suggest the following as the best and most economical arrangement:

| | |
|---|---|
| Rock-breakers | to 2.00 in. |
| Rolls in series crushing wet | " 0.06 " |
| Grinding and amalgamating pans | " 0.02 " |
| Tube-mills | to infinity. |

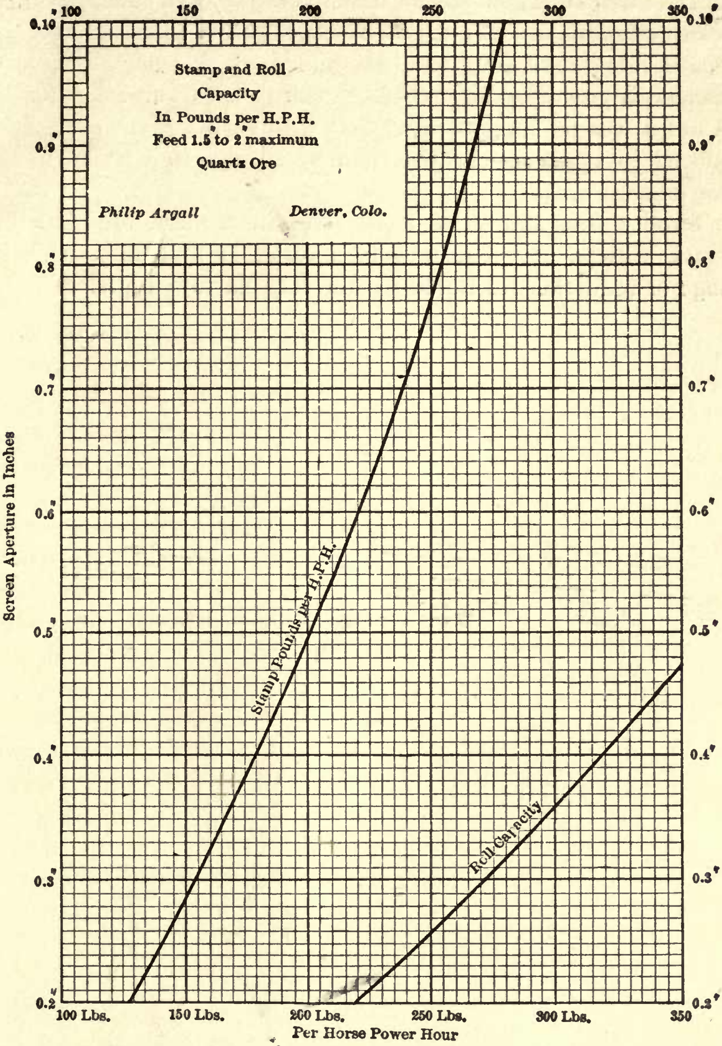

Stamp and Roll
Capacity
In Pounds per H.P.H.
Feed 1.5" to 2" maximum
Quartz Ore
Philip Argall
Denver, Colo.
100
150
200
250
300
350
0.10"
0.9"
0.8"
0.7"
0.6"
0.5"
0.4"
0.3"
0.2"
Screen Aperture in Inches
Stamp Pounds per H.P.H.
Roll Capacity
100 Lbs.
150 Lbs.
200 Lbs.
250 Lbs.
300 Lbs.
350
Per Horse Power Hour

FIG. 16.

The pan-mill might be depended upon to catch most of the amalgamable gold; plates would be required only after the tube-mill, to catch such gold as had been liberated by the finer grinding. I can even conceive of such an arrangement of plant for amalgamation only. Stopping the tube-mill work at such a degree of fineness as gave the best results — call it for argument 1-100 in. — and it can be demonstrated that with such a plant ore can be reduced to 1-100 in. cheaper than by stamps to 1-50 in. in the best modern mills.

Finally, the stamp, if it survives in modern ore-reduction plants, will be only one unit in a series of machines, with its range of reduction restricted within very narrow limits.

## SPRINGS ON CRUSHING ROLLS

By Lewis Searing

(April 6, 1905)

Most machinery peculiar to any one industry is the result of an evolutionary process involving experiment and the test of time; nevertheless, some useless and complicated feature of design may remain, simply because its existence has never been questioned. Such machines are designed according to tradition and custom, without reference to the particular necessity of each case. The use of springs on the so-called modern ore-crushing rolls is, in my opinion, one of the useless features of design which has not heretofore been questioned.

The use of the springs is a provision for but one condition of operation — the presence of foreign uncrushable material in the ore. There can be no doubt that sudden strain is developed in the rolls when a hammer-head, or similar article, is caught between the shells; but whether the springs are absolutely necessary to prevent serious damage under this condition is a point seldom considered by the mill superintendent; rather, he takes it for granted that they are necessary; indeed, it has been well said that the use of springs is a habit established by estimated, rather than actual, necessity.

The disadvantages of the spring rolls are twofold. In the first place, a set of good spring rolls is costly in design and in manufacture; 15 to 20 per cent. of the cost can be traced directly to the springs and their attendant mechanism. In the second place, any neglect of the spring tension will result in an undue proportion of oversized product. Who can say that it is possible to secure an adjustment of the springs which will produce an even product and at the same time permit of sufficient flexibility for the passage of a hammer-head without throwing the driving-belt?

The absolute prevention of an oversize product is greatly to

be desired when such a result can be obtained, and this is possible by the use of rigid rolls. It materially reduces the time and cost of crushing, and also eliminates one elevator and its accessories, and permits a reduction in either the size or the number of rolls to be installed.

A minor point, worthy of consideration, is the pounding, jarring, and vibration accompanying the operation of spring rolls — hard upon the building and attendants alike — but which is entirely obviated by the use of rigid rolls.

It is well known to mechanical engineers that the shock and strain on rolls employed in the manufacture of steel rails is terrific, especially when the steel billet is introduced to the first, or breaking-down, set of rolls. It requires an enormous amount of power to reduce the thickness of the metal a fraction of an inch — enough, in fact, to produce a perceptible momentary reduction in the speed of the huge engines employed, the change from no load to full load being instantaneous. Again, the tremendous variation of load on steam shovels, dredges, and similar machines is a matter of common knowledge. There is also the case of the ore-crusher. Notwithstanding the strain and shock to which this type of machinery is subjected, no springs are involved in the mechanism, and the design shows all parts fixed and rigid throughout.

Now, if the complications attendant upon the use of springs are avoided in steel rolling-mills, steam shovels, and dredges, and the various forms of ore-crushers, why is it not reasonable to affirm that these complications can also be avoided in the construction of crushing rolls, which, it must be allowed, have comparatively easy duty? If these other machines can be made to operate successfully under far more severe duty, then ore-crushing rolls involving the same principle of design will also operate successfully.

Even if there were no difference in the first cost, and the entire extra cost of the spring mechanism were devoted to strengthening the bearings and frame of rigid rolls, the simplification of the machine and the prevention of oversized product would be accomplished. As a matter of fact, rigid rolls are now being manufactured and sold at appreciably lower prices than spring rolls of the same size. If rigid rolls be properly designed it is impossible for the belt to transmit sufficient power to injure the

rolls; but, as a precaution, a safety device, such as a wooden, instead of a metal, key in the driving pulleys may be used. In the event of complete stoppage, this key will sheer, or the belt will slip, before the rolls can be injured.

It may be argued that rigid rolls would be acceptable were it not necessary to provide an adjustment to compensate for wear. This, however, is a simple matter and as easily arranged as the adjustment necessary on the rolls in the steel mill.

Finally, in view of the fact that rigid rolls have been in successful operation at the plants of the Osceola Copper Company, at Houghton, Michigan, and the North American Lead Company, at Fredericktown, Missouri, it would seem a safe prediction to say that the use of springs on crushing rolls will soon become a matter of interesting mechanical history.

# NOTES ON SOME REGRINDING MACHINES

By Martin Schwerin

(March 10, 1904)

Two years ago I made a series of tests on four types of regrinding machines in the old concentrator of the Anaconda Copper Mining Company, at Anaconda, Montana. The results of these, together with some observations, are here given in the belief that the interest in this feature of ore-dressing is unabated. That the difficulties and complexities have not diminished is attested by the divergence of opinions held by recent writers.

In any comparison of fine grinding machines and reports of work accomplished by them the condition of the ore fed into them is recognized to have a significant effect. It is evident that the condition of the ore — whether it is fine or coarse — not only affects the capacity and character of work done, but even when these conditions are the same for all, various machines are affected differently by changes in the conditions.

In the published reports of ball-mills working a coarse material from a Blake or Gates crusher and discharging it in a finely comminuted condition, it appears that the machine is well adapted to the work. It is a most efficient screening machine, and on this account it is better adapted to a preliminary reduction of ore in preparing it for a first metallurgical treatment, than it is for regrinding particles that are already reduced to a size approaching that desired in the discharge of the machine; more concisely, it is better adapted to grinding between very wide size-limits than it is for regrinding between very close size-limits.

In the latter case it frequently happens that as much as from 50 to 75 per cent. of the feed is already smaller than the width of slot or free aperture of the screen-cloth used in the machine. This was the case at Anaconda, where the feed was wet jig-tailings. The result, in this use of the ball-mill, was a rapid screening out of the under-size feed with scarcely any grinding. It may be

suggested that the remedy is the use of a finer screen in the ball-mill; but this is not the remedy. Practical considerations and the necessity for discharging a jigable product control the size of the screen and do not allow the use of one with much less than 1.5 mm. slots. It is absolutely essential that any successful regrinding machine should be able to work effectively between close size-limits.

In the appended tabulated results it is seen that the screening ability of the ball-mill and the deterioration in effective crushing increase alarmingly with the life of the lining and screens. This is partly due to the enlargement of the holes in the armor-plate and partly to the wear of the screens. From a diameter at the small end of the holes in the armor-plate of $\frac{9}{16}$ in. they wore in five months to $\frac{7}{8}$ in. The increased size afforded greater facility for the particles of ore to pass through to the outer screens, thereby escaping the action of the balls. While a granular product is desirable, it must not be attained at the sacrifice of effective grinding.

The tailings from jigs handling the ball-mill product for a period of ten days, during which they were systematically sampled, averaged 0.925 per cent. copper. During the same period the tailings from jigs treating the product from Chilean mills averaged 0.595 per cent. copper. The feed of both machines was nearly the same in copper value. During the same period the tailings from the first tail-sieve of the ball-mill jigs (treating the coarsest of four sizes made by the hydraulic classifier) assayed 1.3 per cent. copper.

Such results are to be expected from an inspection of the asasys on the sized products of the two machines. From these we see how much richer in proportion to the assay of the unsized feed are the coarse sizes discharged by the ball-mill compared to the coarse sizes discharged by the Chilean mill. Bearing in mind that the feed is tailings and therefore contains very little, if any, free mineral in the coarse sizes, it is evident that the high assay of those sizes from the ball-mill is a consequence of the failure of the machine to grind them. These particles leave the machine containing the locked-in mineral, to free which was the object of regrinding. That such enclosed mineral goes into the fine sizes when released is shown by an inspection of the assays of the material discharged by the Chilean mill. It is evident that the

ball-mill is not adapted to regrinding tailings for further concentration.

Another serious objection to it is that it requires most careful feed regulation when running anywhere near its capacity. When the feed becomes heavier than the machine can handle and continues so for a while, it packs solidly between the screens and the armor-plate, and it becomes necessary to shut down, remove the outside screens, and then rotate the mill to free it. Without shutting down, certain hours of running become necessary, with all feed shut off, to get the machine free again. This is one of the practical considerations, before mentioned, which control the size of the screens. But this objection, like the others, has not the same application when crushing large pieces to a small size, because in this latter case the screening capacity is so greatly in excess of its grinding capacity that no congestion of the screens can occur through the mere mass of material forced against them.

The absence of the dull roar characteristic of the ball-mill when running properly gives warning of the danger from overfeeding. But the world over the "graveyard shift" in a mill is notoriously heedless of all such warnings.

The size of the Krupp ball-mill was that measuring 10 ft. 10 in. in hight by 15 ft. 6 in. in breadth over all, with drum 4 ft. 6 in. by 9 ft. diameter.

The Chilean mill tested was an original Bradley mill, in which certain improvements were made at the foundry of the Anaconda Copper Mining Company, under the direction of Messrs. Evans and Waddel. The improvements consisted principally of a spider with rigid arms on which the rollers were hung in the place of the three-socket jointed trunnions.

This mill is well adapted to regrinding lean tailings and poorly suited for comparatively rich tailings or middlings. Lean tailings from copper concentrating contain but a small quantity of sulphides very thoroughly distributed through the quartz grains as minute enclosed particles. It is necessary, in order to liberate the sulphides, that the whole of the feed be very finely pulverized. And in doing this, if the process be not carried too far, there is but slight danger of the liberated mineral particles being ground a second time, thereby unduly sliming them, because of the protection afforded by the larger, harder, and more resistant grains of quartz. Such material is the tailings from jigs after

fine rolls. It will usually be found more profitable to regrind the middlings from such jigs in the same set of rolls or in a Huntington mill.

The action most desirable on such middlings is to have each grain cracked open once and then freed. Rolls do this almost perfectly, provided the grains are not too small for effective work. Huntington mills do it less perfectly, but are capable of doing effective work on very much smaller grains. This action would not be sufficient for the poor tailings. On them the cracking or grinding must be repeated. The sizing assays show how well the Chilean mill does this. But this very desideratum for the tailings, if carried out on middlings would be disastrous on account of the larger size of the enclosed sulphide particles in proportion to the size of the enclosing grains of gangue.

The Chilean mill has a large capacity, which is a very necessary adjunct of any machine put to regrinding tailings. Since the tenor in copper of the first wasted tailings is influenced no less by the proportion cut out for regrinding than by the character of the regrinding itself, the greater the proportion reground, the poorer will be those tailings sent to the dump.

This machine is not sensitive to varying loads. If over- or under-loaded it does its work without serious impairment of efficiency.

The size of the screens is the principal factor in the capacity of the Chilean mill and may even be said to control it. Being on the outside, they are perfectly accessible, as they are in a stamp-mill, and the condition in which they are kept by the attendant is of great importance if the full duty is to be maintained. The output will be almost proportional to the ratio between the number of blinded holes to the total number of holes. But if the duty of the machine be light, a regulation of the character of the discharged product can be exercised through the condition of the screens. By keeping a large proportion of holes blinded all the time almost any degree of pulverization can be attained. The same end, however, can be accomplished better and more directly by the use of finer screens.

Attention is called to the results of the experiment designed to show the variations in the product corresponding to variations in the load. The first column is not important, but it is given to show that there were differences in the assays of

## VARIATIONS IN CHILEAN MILL PRODUCT CORRESPONDING TO VARIATIONS IN LOAD

| COPPER ASSAY OF UNSIZED SAMPLE % | TONS PER HOUR | SIZE | WEIGHT % | COPPER ASSAY % | DISTRIBUTION OF COPPER % |
|---|---|---|---|---|---|
| 1.49 | 2.2 | over 20 mesh | 9.3 | .96 | 5.9 |
| | | " 40 " | 32.0 | .91 | 19.6 |
| | | " 80 " | 20.0 | 1.15 | 15.3 |
| | | thr. 80 " | 38.5 | 2.30 | 59.4 |
| 1.37 | 2.3 | over 20 mesh | 7.5 | .87 | 4.7 |
| | | " 40 " | 30.3 | .78 | 17.2 |
| | | " 80 " | 22.8 | 1.00 | 16.6 |
| | | thr. 80 " | 39.4 | 2.15 | 61.5 |
| 1.30 | 4.0 | over 20 mesh | 7.7 | .91 | 5.4 |
| | | " 40 " | 38.8 | .86 | 25.7 |
| | | " 80 " | 20.0 | 1.04 | 16.0 |
| | | thr. 80 " | 33.3 | 2.08 | 52.9 |
| 1.39 | 5.0 | over 20 mesh | 13.5 | .75 | 7.3 |
| | | " 40 " | 37.1 | .85 | 22.7 |
| | | " 80 " | 19.1 | 1.09 | 14.9 |
| | | thr. 80 " | 30.3 | 2.54 | 55.1 |
| 1.71 | 6.0 | over 20 mesh | 18.7 | .83 | 9.0 |
| | | " 40 " | 31.8 | .97 | 18.0 |
| | | " 80 " | 19.7 | 1.74 | 20.0 |
| | | thr. 80 " | 29.7 | 3.05 | 53.0 |
| 1.58 | 6.6 | over 20 mesh | 18.8 | .88 | 10.5 |
| | | " 40 " | 35.7 | 1.12 | 25.3 |
| | | " 80 " | 19.6 | 1.52 | 18.7 |
| | | thr. 80 " | 25.9 | 2.78 | 45.5 |
| 1.48 | 7.25 | over 20 mesh | 28.1 | .95 | 18.0 |
| | | " 40 " | 30.2 | 1.07 | 21.8 |
| | | " 80 " | 16.6 | 1.40 | 16.4 |
| | | thr. 80 " | 25.0 | 2.65 | 43.8 |
| 1.48 | 8.16 | over 20 mesh | 26.3 | .75 | 13.3 |
| | | " 40 " | 34.8 | 1.08 | 25.3 |
| | | " 80 " | 19.0 | 1.66 | 21.3 |
| | | thr. 80 " | 20.0 | 2.96 | 40.1 |

### Huntington Mill Product

AVERAGE OF 6 SAMPLES

| | | | | | |
|---|---|---|---|---|---|
| 1.48 | | Over 20 mesh | 12.7 | .82 | 7.0 |
| | | " 40 " | 43.5 | .92 | 27.0 |
| | | " 80 " | 17.8 | 1.52 | 18.1 |
| | | " 160 " | 10.9 | 2.07 | 15.2 |
| | | thr. 160 " | 15.0 | 3.22 | 32.6 |

### Krupp Mill Product

AVERAGE OF 5 SAMPLES (DURING 1st MONTH'S RUN)

| | | | | | |
|---|---|---|---|---|---|
| 1.98 | | Over 10 mesh | 3.4 | 1.60 | 2.7 |
| | | " 20 " | 20.3 | 1.65 | 16.9 |
| | | " 40 " | 37.6 | 1.54 | 29.2 |
| | | " 80 " | 15.3 | 1.83 | 14.1 |
| | | " 160 " | 13.5 | 2.70 | 18.4 |
| | | thr. 160 " | 11.4 | 3.21 | 18.4 |

| | | | | | |
|---|---|---|---|---|---|
| | | **Krupp Mill Product**<br>AVERAGE OF 4 SAMPLES (DURING 6 MONTHS' RUN) | | | |
| 1.4 | | Over 10 mesh | 23.7 | 1.41 | 22.9 |
| | | " 20 " | 31.7 | 1.35 | 29.3 |
| | | " 40 " | 29.5 | 1.28 | 25.8 |
| | | " 80 " | 5.5 | 1.52 | 5.7 |
| | | " 160 " | 4.4 | 2.16 | 6.5 |
| | | thr. 160 " | 4.4 | 3.36 | 10.1 |
| | | **Gates Roll Product**<br>AVERAGE OF 4 SAMPLES | | | |
| | | Over 10 mesh | 14.2 | | |
| | | " 20 " | 30.2 | | |
| | | " 40 " | 29.6 | | |
| | | " 80 " | 12.2 | | |
| | | " 160 " | 5.8 | | |
| | | thr. 160 " | 7.3 | | |

## SUMMARY

| | KRUPP BALL-MILL NO. 8 | HUNTINGTON MILL | CHILEAN MILL | GATES ROLLS |
|---|---|---|---|---|
| Capacity in tons per hour | 5.0 | 3.0 | 6.0 | 9.0 |
| Screens used (mm.) | 1.5 | 1.5 | 1.5 | 1.5 |
| Speed r.p.m. | 21 | 65 | 34 | 108 |
| Size of machine | DRUM 4 FT. 6 IN. BY 6 FT. DIAM. | DIE RING 5 FT. DIAM. | DIE RING 6 FT. DIAM. | SHELLS 15 IN. FACE BY 36 IN. DIAM. |

the feed. Such differences were unsought and undesirable, but were unavoidable. From the table it is seen that, as the tons per hour increase, the discharge contains greater percentages of coarse sizes and smaller percentages of fine sizes. This is explained when it is remembered that the greater the quantity of material in the machine the greater is the amount that is dashed against the screens at each revolution of the rollers and scrapers, affording opportunity for greater numbers of particles to escape without grinding or with less grinding, while the amount of material that is actually ground between the rollers and die-ring is not increased. Also, since there can be no accumulation of feed beyond a certain point, soon reached, the greater the amount fed into the machine the shorter is the time a given particle can stay in it.

In the last column are shown the relative amounts of copper

in the several sizes. The lower tonnages, accompanied by more perfect grinding, are also characterized by less copper in the coarse size. This column shows how the slimes run up in copper as the burden on the machine diminishes, and how the coarse sizes carry more as the burden increases. These effects are apparent when the extreme sizes are used as criteria. No information is furnished, apparently, by the intermediate sizes with respect to variations in weight percentages, but the copper assays on these sizes indicate that they are leaner when the amount of feed is small and richer when the amount of feed is large.

The more copper in the coarse sizes (using the last column as the criterion) the less favorable is the product for concentration, bearing in mind, of course, that these are lean tailings in which nearly all of the copper minerals are in very fine particles imbedded in gangue from which a concentrate of the coarser sizes cannot be made. The results are believed to substantiate the statement that the Chilean mill finds its proper place in regrinding the leanest practicable tailings. If the feed were richer and the individual sulphide particles larger, the effect of sliming would become prejudicial.

Some further elucidation becomes necessary to a complete understanding of the sizing assays. Although the facts stick out plainly enough, the reason is not at once manifest why the same coarse size in the products of different machines shows so great a difference in copper contents, even after making any correction that may be thought proper on account of the initial assay differences between one unsized discharge and another. The mere fact that in the ball-mill discharge there is a much larger percentage of the size designated as over 20-mesh than in the Chilean mill product is not of itself good reason why it also assays higher. The like sizes in the two products contain unlike material. In the case of the ball-mill the size under discussion contains nearly the identical grains of that size which came in with the feed, while in the case of the Chilean mill product the same size contains a greater proportion of grains newly made from the breaking up of larger grains in the machine. And so with the next smaller size to a less extent. When a large grain is broken it is approximately true that the sulphides find their place way down the table of sizes, while the quartzose material stays nearly where it was in the scale. The grains consisting partly of sulphides and partly

of gangue take intermediate positions corresponding to the predominating component mineral.

The Huntington mill tested was one of the under-driven, geared, 5-ft. size, and was run at 65 revolutions per minute. The product made is seen to be similar to that of the Chilean mill when handling about 6.5 tons per hour. From this it must not be inferred that the results would be so nearly alike on material a great deal richer, such as middlings might be. On the richer feed the Huntington mill results would not change greatly with respect to distribution of copper, while the Chilean mill product would correspond, with respect to copper distribution, more nearly to the results shown for a burden of 2 tons per hour, which, although an actual necessity on the low-grade tailings, would be unnecessary and highly undesirable on rich stuff because the slime losses would be excessive in the subsequent operations. On such stuff, therefore, the Huntington mill shows to great advantage.

With this mill, as with the Chilean mill, screens of any desired degree of fineness can be used, but in my opinion whenever it becomes necessary in concentrating to pulverize very completely material with a hard, tough gangue, that duty should be assigned to a Chilean mill. Wet crushing between close size limits is premised in every case.

A fault of the Huntington mill is the uneven wearing of the roller rings. Instead of wearing evenly to successively smaller concentric circles, they frequently assume polygonal shapes and then pound around the inside of the die ring instead of rolling in continuous contact with it. When this fault becomes aggravated the grinding falls to a fraction of the normal capacity, and the machine becomes a deafening rattle-box. The new 6-ft. mills running at 52 revolutions per minute in the west half of the new mill at Anaconda show a very considerable increase in capacity over the smaller ones. They run more smoothly, wear the roller and die rings more evenly, and are a considerable improvement in every way.

The rolls tested were Gates' heavy pattern, with shells 15 in. by 36 in., run at 108 revolutions per minute, giving a peripheral speed of 1018 ft. per minute. The usual arrangement of rolls and trommels was used, in which the feed first passes to an elevator, then to the trommels, the under-size going to jigs and

the over-size to the rolls. After passing the rolls the crushed material joins the incoming feed and goes up in the elevator to the trommels again.

The most conspicuous feature of the roll product is the small quantity of slimes present. When the minerals which will make a desirable concentrate can be set free by crushing the containing tailings to pass 1.5 or 1.25 mm. there is no machine so well adapted to do it as rolls. On such material the recovery of concentrates after roll-crushing exceeds the recovery following any other machine. Their capacity is far in excess of any other regrinding machine, crushing to the same size. But the lower limiting size is reached here, unfortunately. The mechanical difficulties present in the rolls themselves, which lower their effectiveness when crushing to a finer size, as well as the impracticability and low efficiency of trommels using a finer cloth, has hitherto prevented their use for finer grinding.

However, the day is confidently looked for when rolls will supersede the Huntington mill, which is now the machine best suited to the regrinding of material to sizes finer than 1.25 mm. when the avoidance of slimes is a prime necessity. Hydraulic classification may take the place of the trommel in the roll system and return the coarse size to the machine, which with improvements in the rolls themselves, in the metal of the shells and in methods of distributing the feed so as to cause more even wear, will undoubtedly enable rolls to do finer work than they are now assigned to.

## REGRINDING MACHINERY

By S. V. Trent

(March 31, 1904)

I have read with much interest the statement of tests made by Martin Schwerin in the *Engineering and Mining Journal* of March 10. Having given a good deal of study and observation to the matter of crushing and pulverizing machinery, I am naturally interested in a subject of this kind, especially as I happen to be familiar with the operations at Anaconda.

While I believe Mr. Schwerin's conclusions are in the main sound and correct, I believe his generalizations in regard to Chilean mills, Huntington mills, rolls, etc., are very far from final, and that he himself will be willing to admit this, after consideration of certain features that he entirely overlooks in the article in question. Furthermore, there is one all-important feature developed by the earlier tests at Anaconda of which he makes absolutely no mention. I refer to the fact that in the comparison between the Monadnock mills that were used in the old plant of the upper works, as compared with the performance of crushing rolls, Gates and others, doing similar work, it was found that the final tailings from the tables taking their feed from the Monadnock mills showed a much lower residue of copper, ranging from 0.5 to 0.6 per cent. On the other hand, the tailings from tables taking their feed from the crushing rolls of different makes showed an average copper content varying between 0.85 and 0.95. This was indeed a revelation, and could only be accounted for on the supposition of a more favorable form of grinding by the mills, as compared with the simple cracking operation of the rolls that Mr. Schwerin lays such stress upon.

As regards the Evans-Chilean mill, whose work was the basis of the test report referred to, Mr. Schwerin overlooks the fact that it was proportioned with extremely bad judgment for the work to be accomplished. Mills of the Chilean type, when pro-

portioned intelligently for the required work, are capable of almost infinite variation. If a pulverizing machine proportioned for handling the hardest of rock is used upon soft or friable quartz, unsatisfactory results must of course be anticipated, in the same way as improper results will accrue where pulverizing machinery adapted for soft ore is used upon the hardest. What I mean to say is this — that, with the Chilean form of mill properly proportioned, the percentage of fines is very largely under control.

Of course, the longer material remains in a pulverizing machine, the more finely it will be pulverized, the supposition being that a machine is at all times receiving a feed properly calculated for its capacity. Where a properly designed Chilean mill of large capacity is called upon to do its work on a reduced capacity, it is simply a question of reducing the speed of the mill to get a reduced capacity without producing excessive fines, and with a crushing force properly proportioned to the hardness and size of the feed, the proportion of fines can be kept under good control.

Furthermore, I must take exception to Mr. Schwerin's dictum that "wet crushing between close size limits should be premised in every case." Using the form of Chilean mill with which I am particularly identified, it is entirely feasible, even with very hard rock, to take a feed of up to 2 or 3 in., and reduce to anywhere from 20- to 60-mesh at the one operation, without any material detriment to the machine.

Again, referring to that paragraph concerning crushing rolls, from which I quote the following: "The most conspicuous feature of the roll product is the small quantity of slime present." Of course, where a machine does not pulverize, but simply crushes, as in the case of rolls, as much slime may not be produced as from a strictly pulverizing operation; but if the copper remains in the final tails, what practical advantage is there in the suppression of fines?

I also wish to challenge this further statement: "Their [crushing rolls] capacity is far in excess of any other regrinding machine crushing to the same size." Taking a Chilean mill of the size and weight that Mr. Schwerin refers to, it is entirely feasible to arrive at a pulverizing capacity of 12 tons per hour, pulverizing through 1.25 mm. screen, as against the 9 tons per hour that Mr. Schwerin quotes as the capacity of the 36 by 15 in. rolls. From Mr. Schwerin's table of tests it would appear that the

extreme range of the Chilean mill rises to 8 tons per hour, but that is by no means the limit.

The auxiliary machines incidental to crushing rolls on this class of work, comprising elevators and fine screens, is a matter that Mr. Schwerin passes over lightly, but they are serious features, especially in a plant of large capacity.

## REGRINDING MACHINERY

By George E. Collins

(April 21, 1904)

I have been greatly interested in the valuable notes contributed by Mr. Schwerin under the above heading, and from my own experience can endorse most of his conclusions.

Rolls are sometimes preferable, because of the minimum proportion of slimes produced; but their capacity is small when fine crushing — under 20- or 30-mesh — is necessary. Moreover, unless the entire product is made to pass through a screen of finer mesh than the smallest particles of the feed, the same objection which Mr. Schwerin raises to Krupp mills for regrinding for subsequent concentration applies to rolls in even greater degree — that much of the material passes through without any recrushing at all.

Huntingtons slime less than Chilean mills, and for a friable ore are preferable; but with a hard and tough gangue they often fail to pulverize, making a large proportion of rounded gravelly particles.

The Chilean mill — of which I have had experience only with the Monadnock type, built by Messrs. Trent — is the most widely useful machine I know of for moderately fine regrinding. One of its drawbacks, to which neither Mr. Schwerin nor Mr. Trent alludes, is the variation in product and output as die and rings wear down: in the former case owing to increasing hight of discharge, as in a stamp-mill; in the latter because of lessened weight of mullers. This variation cannot be closely controlled by changing the screen-mesh; and is only partially obviated by care in keeping the scrapers close to the die.

The same point which Mr. Schwerin notes with Huntington mills — irregular wearing of rings and dies — often applies in less degree to Chilean mills. I agree with Mr. Trent that the Monadnock mill can successfully reduce comparatively coarse

feed to any desired mesh at one operation, without proportionately reducing capacity; so can the Huntington. I doubt, however, whether 2- to 3-in. sizes can be fed with advantage, as this results in undue strains on running gear and foundations.

I am disposed to think that the importance of gradual reduction — the cracking of each grain once only — is often overrated. The carrying of this ideal into practice sometimes results in the circulation round and round, in trommels and elevators, of a great bulk of middlings, the more friable (and usually the richest) constituents of which are often literally worn away. It will frequently pay better to recrush a little finer, at one stage.

Similar experiments to those of Mr. Schwerin in really *fine* grinding — below 100-mesh — of hard ores would be valuable. In the case of the Cornish tin-ores, for instance, much of the mineral is often not liberated without very fine grinding. For this work, in my time, the Stephens-Toy pulverizer was largely used, a machine obviously modeled on the pans used in pan-amalgamating. This machine held the field, notwithstanding its heavy costs in power and repairs. For such extremely fine grinding, there was obviously reason for the local use of discharge by "flush" in place of screens. Possibly the Gilpin county burr-slot screen would have answered.

In some recent laboratory experiments in the concentration of a wolfram ore, I found that only a very imperfect separation could be effected without crushing to 80-mesh; and even after regrinding to 100-mesh there was a considerable proportion of middlings which needed still finer grinding. Nothing short of the regrinding such as is done by the Cornish "streamers" on the Red River and elsewhere would have liberated this mineral so as to permit of a clean product being obtained.

# THE FERRARIS BALL-MILL

By W. R. Ingalls

(November 28, 1903)

The Ferraris mill is a new device of the ball-mill type, especially adapted for wet-crushing, which is now in successful use at the works of the Societá di Monteponi, at Monteponi, Sardinia, and at the works of the Société Minière du Gard at Durfort, France. At both places it is employed for the fine crushing of mixed sulphide ores as a preliminary to separation on shaking tables. Other installations besides those mentioned are shortly to be made. As a machine for fine crushing the mill is especially interesting because of its simplicity, and therefore its comparatively low cost. It is the invention of Signor Erminio Ferraris, the director-general of the Societá di Monteponi.

The general construction of the mill is shown in the accompanying engravings. A steel drum is divided into two compartments by a perforated annular partition. The larger compartment, into which the ore is fed, is lined with hard steel plates, and contains the usual balls for crushing the ore. The smaller compartment is divided into a series of pockets by means of a cone, protruding into the larger compartment, and a series of radial partitions extending thereform. The ore, having been crushed to a certain degree in the larger compartments, escapes through the holes in the partition into the smaller compartment. If it has been crushed to the desired fineness, it is discharged through the screens in the end of the drum; otherwise, it is elevated in the radial pockets of the smaller compartment, by the rotation of the drum, until it slides back on the surface of the cone into the larger compartment, where it undergoes further crushing.

The steel drum is supported on four bearing rollers, which are driven by suitable mechanism, turning the drum by friction around its longitudinal axis. The tires are made of cast steel

and the rollers of chilled cast iron. The shell of the drum is made of ⅜-in. wrought iron or steel plate, and the drum is divided by an annular perforated partition into the two compartments above described. In the mills in use at Monteponi the larger compartment is about 56 in. in diameter and 33 in. in length. About 800 lb. of balls are used therein, the balls varying in size from 6 in. diameter down to 3 in. It is necessary to have balls of different sizes in order to effect the crushing. The balls must be made of forged steel; if made of cast steel without forging, they will not remain spherical as they wear down in use. The

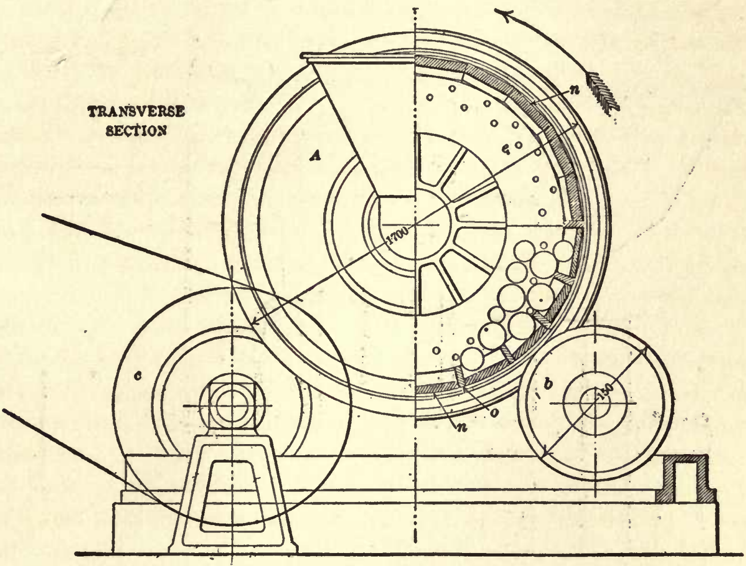



FIG. 17. — Ferraris Ball-Mill.

crushing compartment is lined with bars of cast steel, 1.5 in. thick, with projecting ribs. The form of the ribs need not be precisely as shown in the drawings; under certain circumstances, bars of trapezoidal form might be preferable. The lining need not be adjusted precisely, because the crushed ore will fill the spaces. At Monteponi the rough bars are put in just as they come out of the mold.

The smaller compartment of the drum is about 8 in. in length. The peripheral holes in the dividing partition are 0.6 in. in diameter on the side of the larger compartment, and 1 in. in diameter

on the side of the smaller. The size of these holes should vary according to the kind of ore to be crushed. If the ore has an especial tendency to slime, it will be advisable to make the holes larger than otherwise in order to let the material more quickly out of the crushing compartment, giving the fine stuff an opportunity to pass out through the delivery screens, the larger pieces being returned to the crushing compartment for further reduction.

The cylinder is rotated at 20 r. p. m. They have been operated at 16 and at 30 r. p. m., but their efficiency has been

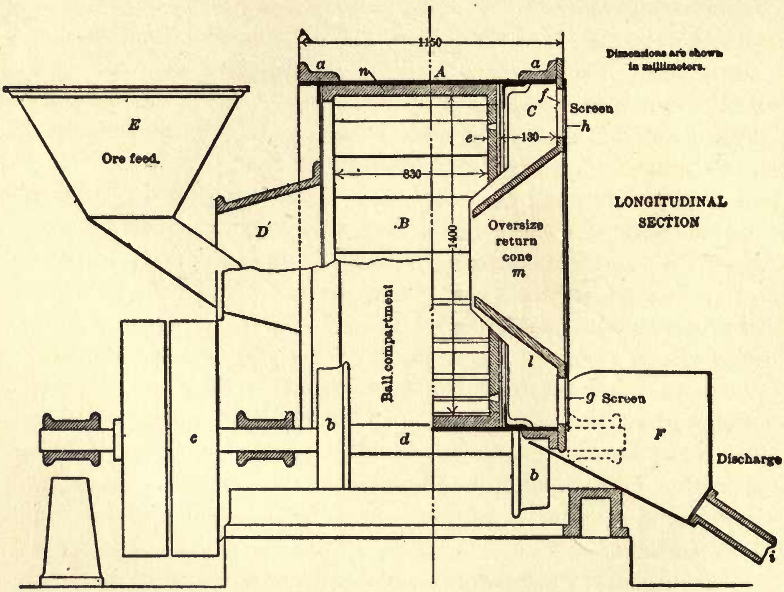



Fig. 18.—Ferraris Ball-Mill.

proved to be greatest at 20 revolutions. The mill has crushed 4000 kg. of quartzose ore to pass a 3-mm. screen in three hours, the feed being in lumps. This is 8818 lb. or 2939 lb. per hour, or, say, 1.5 short tons, equivalent approximately to 30 tons per day of 20 hours. In the case of a sandstone finely impregnated with blende, fed both in lumps and smalls, the capacity was 1250 kg. per hour, crushing to 1 mm. size.

The advantages of ball-mills as a type of crushing-machine are that they combine in one apparatus the means for pulverizing

and screening the ore and the return of the oversize; at the same time it is a machine which occupies relatively little room, either superficially or in hight, receives ore in large pieces and reduces it to fine size in one operation, and consumes comparatively little power per ton of ore. Its chief disadvantage is the large consumption of steel through wear of the balls and the lining, which is not, however, so large as to offset the advantages. In the Ferraris mills at Monteponi this wear amounts to 100 kg., or 220 lb. in a month, working 10 hours daily. In crushing 1.5 tons of ore per hour to 3 mm. size, this would figure out to about 0.6 lb. of steel per ton of ore.

Contrary to the belief of many who are not practically familiar with the ball-mills as a type, they make a granular product, not a slimy one. The Ferraris ball-mill is designed especially for wet-crushing, and is used very advantageously in connection with bumping or shaking tables for the concentration of ores that require grinding to 1-mm. size (about 12-mesh) or finer. It is used in this way at Monteponi and at Durfort, France. Screens being entirely dispensed with and belt elevators and power-transmitting mechanism being reduced to the minimum, the cost of plant is much reduced, there being also a saving in the building, which can be made comparatively small in floor area and of little hight. Thus a concentrating plant of, say, 50 tons capacity per day can be obtained by installation of one 9- by 15-in. crusher, one short belt elevator to deliver into the ball-mills, two ball-mills, one hydraulic classifier and four Wilfley, or similar tables, and a 40- to 50-horse-power engine. The expense for operation is also reduced to the minimum, since the ball-mills require very little attention.

As compared with stamps, the ball-mill is the more economical. This is shown by the results of certain tests reported in the *Engineering and Mining Journal*, Nov. 9, 1901, page 602. A five-stamp battery with a stamp weight of 525 kg. (about 1.165 lb.), falling 160 mm. (about 6.5 in.), with 92 drops per minute, crushed approximately 780 kg. (about 1720 lb.) of hard quartz per hour through a 40-mesh screen, requiring 12.5 horse-power. A No. 3 Krupp ball-mill crushing wet, with 450 kg. of special steel balls, passed 1200 kg. (about 2650 lb.) of the same quartz through the same screen per hour, using only 9 horse-power. Of the material crushed by the stamps to pass a 40-mesh sieve, only 6.5 per cent.

was retained on a 60-mesh sieve, while in the case of the ball-mill product 32.6 per cent. would not go through a 60-mesh sieve. Of the product of the stamps, 56.5 per cent. was fine enough to pass a 150-mesh sieve, as compared with 33.8 per cent. of the ball-mill product.

# THE OPERATION OF A TUBE-MILL [1]

By Hermann Fischer

(November 17, 1904)

The operation of grinding in a tube-mill is carried out in a closed receptacle; one can only see the incoming and outgoing material and hear the noise in the interior of the revolving drum. As for the rest, one must fall back upon a comparison with known operations of an analogous character. For this reason, the opinion has been generally held that the balls roll on the slope of the drum, inside the whirling mass, crushing the ore between them, until it passes from the feed to the discharge at the lower end of the drum.

By means of special arrangements, the firm Fried. Krupp A.-G. Grusonwerk, of Magdeburg-Buckau, have now rendered it possible to observe the operations in the interior of the drum. These observations prove that the explanation hitherto accepted is fundamentally wrong, that the tube-mill does not appreciably grind the ore, either on the slope or in the interior of the whirling mass, but crushes it by an inclined beating action, and that the higher position of the feed, as compared with the discharge, is of no importance for conveying the material.

For the purpose of experiment, a glass drum was first used; then a larger one was constructed, in which the inspection of the interior of the drum was rendered possible by an exchangeable grating, and, finally, a drum of 1 m. (39⅜ in.) interior diameter, of similar construction, was employed. This was first operated without material, and then with material of different kinds, which would not give off dust, at varying speeds of rotation.

The discharge end of the drum was closed only by a wire grating, Figs. 19 and 20, so that the interior was clearly visible. The drum contained only flint balls, about 60 mm. (2⅜ in.) in

[1] Abstract of a paper in the *Zeitschrift des Vereines Deutscher Ingenieure*, March, 1904.

diameter. The hight of arch of this charge was about 450 mm. ($17\frac{3}{4}$ in.) when the drum was in a state of rest. The drum was then turned slowly by hand, until a few balls began to move on the surface. Fig. 19 represents this stage. Then the speed was raised to 21 and $23\frac{1}{2}$ revolutions per minute; the balls rolled rather slowly down the slope; the hight of the mass increased a little. As the speed increased to 28, 30, and 32 revolutions per minute, the motion on the free side became more lively, and the mass became visibly loose; the hight increased to about 600 mm. ($23\frac{5}{8}$ in.). Fig. 20 is an instantaneous view of the operation when the drum was revolving at 32 revolutions per minute. The balls, which are in contact with the drum below and on the ascending side, are carried along without changing their position relatively to the wall of the drum, until they separate therefrom at a certain hight and describe a distinct curve of projection. The balls further inward likewise do not change their position with regard to the drum when ascending, but a curve of projection is hardly perceptible on the descending side. On comparing Fig. 20 with Fig. 19, it will be seen that the contents have become looser. As the speed accelerates to 35 revolutions per minute, this tendency is more marked, so that the hight increases to 650 mm. ($25\frac{5}{8}$ in.). The

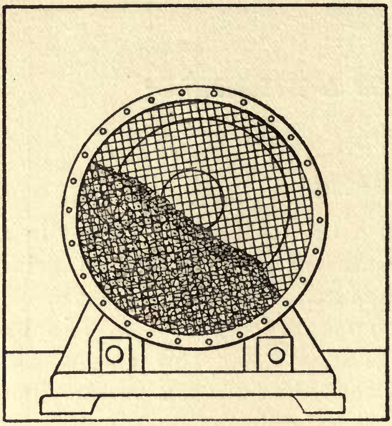


**Fig. 19. — Beginning of Rotation. 1 Meter Diameter.**

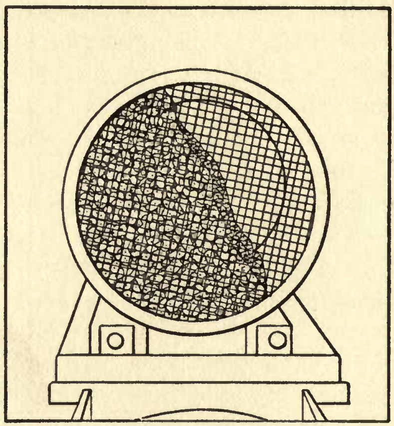


**Fig. 20. — Mill 1 Meter Diameter. 32 Revolutions.**

curve of projection of the balls, which were raised when in contact with the drum, is clearly visible, as is also the fact that the balls further inward are separated in layers. On the ascending side, about 200 mm. ($7\frac{7}{8}$ in.) from the wall of the drum, a few balls roll in a hollow space of oval cross-section without intermingling with those moving further onward. The hollow spaces between the several layers on the descending side are visible throughout, while on the ascending side the layers are close to each other. Figs. 21 and 22 are instantaneous views, obtained with a drum, 300 mm. ($11\frac{13}{16}$ in.) in diameter and revolving at 59 and 66 revolutions per minute. A comparison shows clearly how the looseness of the contents increases with the speed. If the speed of rotation is further increased, the balls form a solid ring, revolving with the drum; a shifting of the balls is not perceivable. Fig. 23 is an instantaneous view, when the drum is revolving at a speed of 55 revolutions per minute.

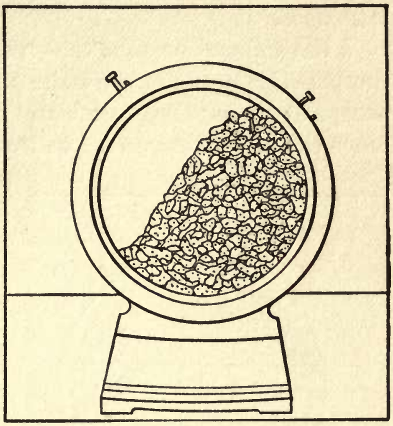


Fig. 21. — Mill 300 mm. Diameter. 59 Revolutions.

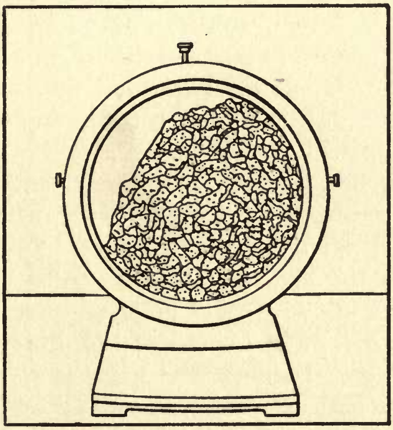


Fig. 22. — Mill 300 mm. Diameter. 66 Revolutions.

A large quantity of material, producing no dust, was now introduced. This material, moving in the same manner as the balls, as was anticipated, entered the hollow spaces between the balls, and ascended and descended with them. A difference was only noticeable in so far as the precipitated layers were less sharply marked,

and the particles of material were spurted laterally, especially at that spot which was hit by the mixture rushing down like a water-jet. Hardly any material was noticed in the contracted oval hollow space.

I have already stated that a sliding or rolling motion between the balls, or between the balls and material, does not take place except at the striking point and in the oval-shaped hollow space. Since the latter is almost free from material, a crushing action — at least, one of any importance — can take place only at the striking point, where the falling balls are violently forced upon the material between them and the balls which have been previously operating. The former play the part of the shoes in a stamp-battery, and the latter act as a substitute for the dies. The crushing is effected by stamping or beating. The action, however, differs from that in stamp-batteries, in so far as the horizontal motion of the falling balls is opposite to that of the balls revolving with the drum.

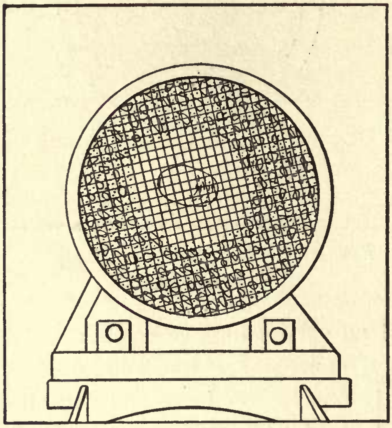


Fig. 23. — Mill 1 Meter Diameter. 66 Revolutions.

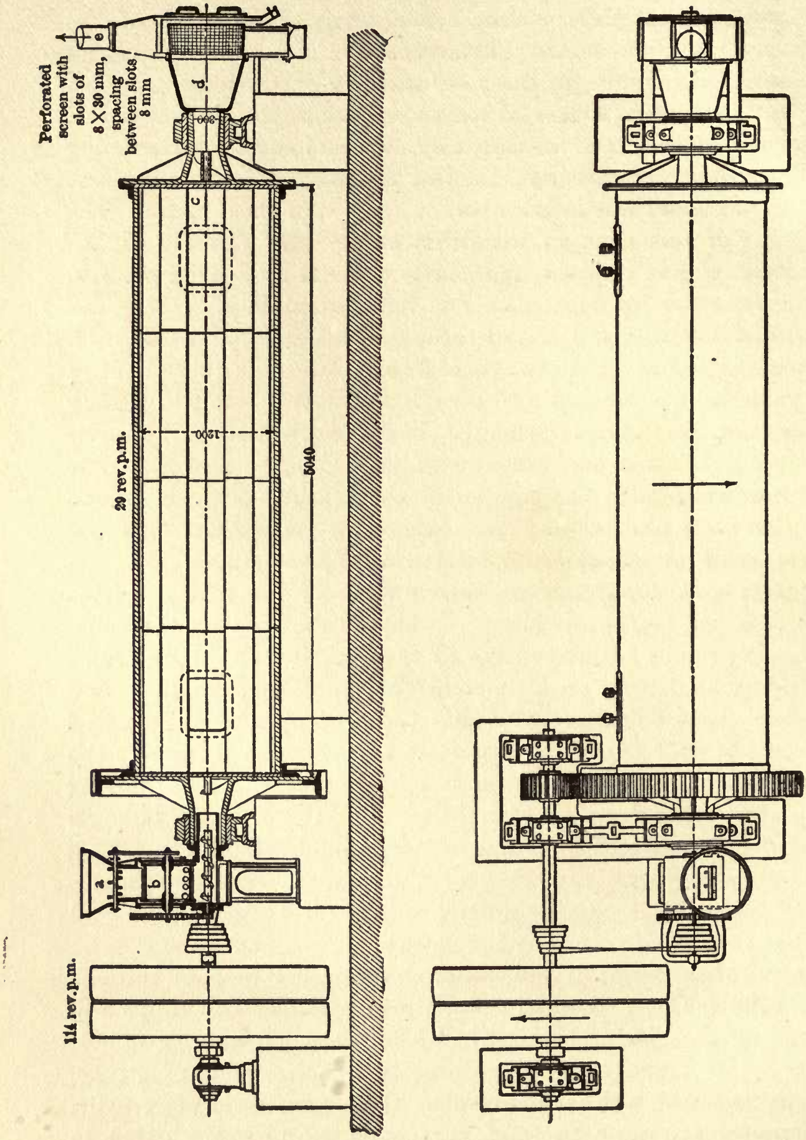
Owing to the force of the strokes, the balls near the striking point will undergo a certain displacement, which, however, can hardly produce a crushing action of any importance. The crushing action thus depends on the hight of fall of the balls — that is, on the hight of the vertex of the path of projection over that point where the balls strike — on the speed of the drum, and on the weight and the number of balls. The speed of the drum must be so chosen that the paths of projection can be well developed. Weight of balls and hight of their fall supplement each other, in so far as they are factors of a product. Harder material requires heavier balls, or a greater hight of fall than softer material, and steel balls can have the same efficiency in smaller drums as flint balls in larger drums. The larger the quantity of balls acting on a certain quantity of material, the

better the crushing. If more material of the same kind is to be crushed to the same degree of fineness within a certain period, the quantity of balls — and, consequently, the length of the drum — must be greater. The respective numerical values can be obtained only on the basis of extensive experiments.

In the further course of the experiments, the discharge end of the drum, Fig. 28, was closed by means of sheet metal, having in its center a trellised opening of 200 mm. ($7\frac{7}{8}$ in.) diameter. To the feed-end was fitted a hollow cone, *c*, of sheet metal. The opening of this cone on the drum side — 500 mm. ($19\frac{11}{16}$ in.) diameter — was trellised and partly covered by a fixed plate, *e*, secured to the machine frame, as represented in Fig. 27. The material was now introduced through a manhole, and the drum was set in operation at the rate of 32 r. p. m. A further quantity of material was thrown into *c* by hand-shovels. It was readily taken through the free opening of the grating, so that each shovelful rapidly vanished. On the outlet side, the material was discharged through the meshes of the grating, conformably to the curve of projection. *The discharge of the material thus took place at a considerably higher level than the feed-opening.*

Consequently, the progressive motion of the material from the feed-end to the discharge — which is required by the tube-mills (ball-mills without sieves at the curved wall of the drum) — does not depend on a difference of hight between inlet and outlet. It is only necessary that the inlet opening be situated where the contents of the drum are loose enough to receive the material, or where the drum is empty, and that the moving contents of the drum pass by the outlet. The drum acts like an elevator; it lifts the balls and material innumerable times to a considerable hight. In view of the sum of these risings, not even the greatest possible difference in hight between inlet and outlet end would be of any influence.

By what means is the material compelled to pass through the drum? That material upon which a falling ball drops will spurt on all sides and be taken up into the neighboring hollow spaces. If much material is situated on the point of stroke, much material will be distributed therefrom; otherwise, little. Consequently, those parts of the contents which are richer in material deliver more than is returned by the parts poorer in material, whereby the proportions of the mixture are equalized.




FIGS. 24 and 25.

Owing to the great activity with which the contents of the drum rise and fall, it was observed that some balls passed over their path twice during one revolution; this equalization takes place very rapidly. At the discharge end, the material which spurts aside passes through the meshes of the latticed opening, while the balls are retained. Thus, at this side, the mixture will be poorer in grinding material, so that the said equalization is directed hither.

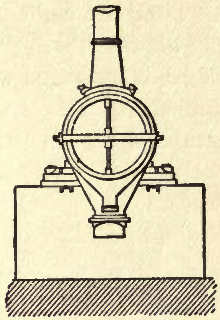


Fig. 26.

Fig. 24 is a longitudinal section, and Fig. 25 a plan of a tube or grit mill; Fig. 26 a cross-section of the drum; Fig. 27 a view of the feed-end, and Fig. 28 a view of the discharge end of the mill. The drum has an interior diameter of 1200 mm. (3 ft. 11¼ in.), an interior length of 5000 mm. (16 ft. 5 in.), and revolves at the rate of 29 r. p. m. The drum consists of sheet iron, 12 mm. (15-32 in.) thick, has cast-steel ends, and is lined with hard cast-iron plates. The grinding balls are introduced through a manhole, while the material to be crushed is fed and discharged through the hollow trunnions

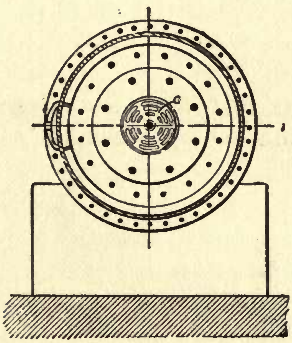


Fig. 28.

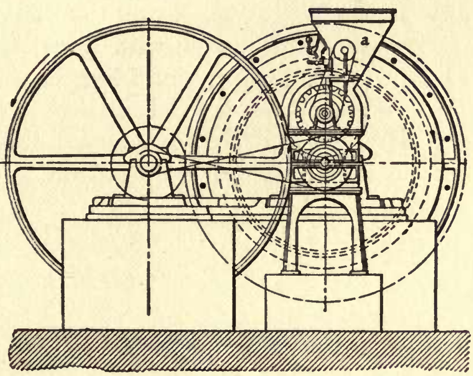



Fig. 27.

cast together with the end walls. The material passes first of all into the hopper, *a*, in which a studded shaft rotates; a longitudinally grooved drum, *b*, regulates the supply. This drum is rotated by toothed wheel gearing and a pair of five-step pulleys at dif-

ferent speeds, and contains a few balls, which serve to shake the drum, and thereby ensure the emptying of its grooves. The material supplied is moved by a screw-conveyor into the hollow trunnion on the left side, Fig. 24, and then passes into the drum, through the latter and through the grating, *c*, into the outlet trunnion on the right side. The inlet trunnion, on the left side, has the blades which prevent the balls from dropping out. The grating, *c*, is provided with arcuate slots of about 25 mm. (1 in.) in diameter. The material passes therefrom into a hopper, *d*, attached to the trunnion. A perforated screen is connected to the hopper. The slots of the screen are 8 mm. ($\frac{5}{16}$ in.) wide and 30 mm. (1 $\frac{3}{16}$ in.) long; they permit the crushed material to drop through, while very hard particles which are not crushed and splinters of flint are retained and subsequently drop out on the right side, Fig. 24. The sieve is enclosed by a casing, out of which the air is drawn by means of the pipe, *e*, so that air enters all permeable parts, thereby preventing the exit of dust. A great advantage of the mills, in which the material is fed and discharged through the hollow trunnions of the drum, consists in the facility of rendering the same free from dust. If, however, the discharge is effected through slots in the drum casing near to the rear end, a comparatively tight closure is not possible, and the non-exit of dust cannot be guaranteed by the suction of air.

These mills are made with a width of 0.95 to 1.5 m. (3 ft. 1$\frac{3}{8}$ in. to 4 ft. 11 in.), and a length of 4 to 8 m. (13 to 26 ft.). The efficiency depends to a large extent on the kind of the material treated and the degree of fineness desired, and, of course, also on the initial size of grain, which is obtained by preliminary crushing.

## THE THEORY OF THE TUBE-MILL [1]

By H. A. White

(September 23, 1905)

When a tube-mill is revolved at a suitable speed, some of the balls at a given moment will be in the air, falling in a certain path, and some will be resting on the linings or on the balls next to the linings. In order to study the motion of the balls, an apparatus was constructed representing a section of a tube-mill. From a thin block of wood, an 8½-in. circle was cut, and was covered front and back with ½-in. plate glass, the joints being sealed with rubber. This was placed in a rotatable holder, and so connected with a motor that any speed might be obtained between 6 and 400 revolutions per minute.

The pulverizing action of the tube-mill is due almost entirely to actual impact of the falling balls in dry crushing, but where the tube is half full of water conditions are modified. A ball falling into two or three feet of water will not strike the bottom with enough force to do much crushing, and in this case it is probable that the grinding action between the balls does a great proportion of the work; the speed of revolution requires adjusting to avoid an unnecessary amount of practically wasted fall, and indeed would be best regulated so that the angle of repose of the crushing material is alone considered.

With these ideas in mind, a series of experiments was made with the apparatus described above, using ¼-in. glass beads to represent the steel balls. The theoretical motion of the balls was calculated trigonometrically, and the calculations verified by experiment, seeking to arrive at the true theoretical motion of the balls. Fig. 29 shows the paths of motion of the balls when the mill is run at varying speeds, permitting them to fall through the air. In this drawing the large circle does not represent the

[1] Abstract of a paper read before the Chem., Met., and Min. Soc. of So. Africa, May 20, 1905.

diameter of a tube-mill, but the diameter of the central or pitch line of an outer layer of balls in a mill. For instance, if a tube-mill were 20 in. diameter inside the lining and 1-in. balls were used, the appropriate circle on the diagram would be 29 in. diameter.

Referring to Fig. 29, a ball that does not receive enough motion to carry it up and send it off at a tangent would have a path designated as zero. On receiving sufficient momentum to carry it to *AII*, it would describe a parabola of 30 deg.; to *AIII* = 35 deg. 16 min., or across the perpendicular center at its lowest point; *AV* = 45 deg., or through the center of the circle; *AVII* = 60 deg.; and so on up to 90 deg., which represents the condition when the balls would cling to the lining continuously, centrifugal force having more influence than gravity.

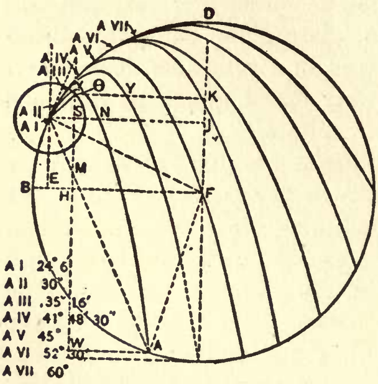



FIG. 29. — Paths of Motion in Tube-Mill.

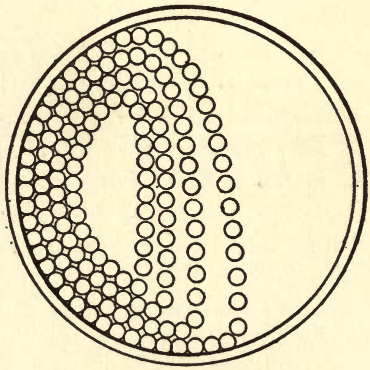


FIG. 30. — Movement of Balls in Tube-Mill.

In practice there are several layers of balls to consider; it is calculated that each particular ball tends to keep to its own layer, and that the theoretical movement of the unimpeded balls is about as shown in Fig. 30. The experiments showed that this tendency of each ball to keep to its layer was approximately true where the mill was not crowded with balls, yet had sufficient to provide the weight necessary to give enough friction between balls and rim to develop the theoretical angle of fall. The proportion existing between the number of balls on the rim and those in the air, at any given moment, may be determined from the time required for each part of the cycle. The balls in the

inner layers have a much shorter cyclic time than those in the outer, or, in other words, having a smaller circle to traverse, at the same speed they come around more quickly.

It is evidently advisable to keep the number of layers small, so that the effective fall may not vary too far from the maximum, but the limit to this is set by the necessity of having enough weight and by consideration of using the full capacity of the tube-mill as far as practicable. If the mill be half filled, the width of layers on the rim cannot exceed 0.293 of the radius and will depend on the angle of departure of the outer layer. An inspection of Fig. 30 shows that interference with falling balls would be caused by too many layers, and this condition should be determined in each case by a diagram.

In the table of diameters and revolutions, on next page, it must be remembered that the diameters in the first two columns are figured from the centers of the balls' paths in their layers, and not from the diameter of the tube-mill. In the third column are given the theoretical conditions when the mill is half full of water, and the revolutions per minute required to keep the balls continuously against the lining. In the fourth column are the revolutions per minute as calculated for dry grinding; and the rotative speeds required to give the curves from *AIII* to *AV* (Fig. 29) are shown in the remaining columns. The diameter of the balls is taken at 1-20 of the circle shown in Fig. 29, and it must be borne in mind that the relative size of the balls affects the result. The speeds found and given in this table closely approximate those given by Hermann Fischer.[1]

I found that the theoretical number of revolutions, even after due allowance was made, required to be exceeded in every case to bring about the desired result. For example, an 8½-in. wooden circle half filled with coarse sand required 94 revolutions in place of 90.9 to make the first layer continuous. Some explanation of this will be noticed in the fact that a single steel or glass ball up to 1½-in. diameter merely revolves on its own axis when placed in my cylindrical section driven at the speed of 400 revolutions per minute. Even 12 steel balls on glass driven at 400 will not rise up the side when the balls are as much as ¼ in. in diameter and the glass 4¼ in. They simply revolve axially and jump a little. It is clear that a sufficient quantity of balls must be

[1] *Zeitschrift des Vereines deutscher Ingenieure,* March 26, 1904.

| D INCHES | D METERS | N HALF FULL; ALL CONTINUOUS | N $\theta=90°$ | N $\theta=45°$ | N $\theta=41° 0' 43''$ | N $\theta=40°$ | N $\theta=35° 15' 51''$ |
|---|---|---|---|---|---|---|---|
| 1 | 0·025 | 297·8 | 265·0 | 222·9 | 214·7 | 212·5 | 201·4 |
| 2 | 0·051 | 210·6 | 187·4 | 157·6 | 151·8 | 150·3 | 142·4 |
| 3 | 0·076 | 171·9 | 153·0 | 128·7 | 124·0 | 122·7 | 116·3 |
| | 0·100 | | 133·6 | | | | |
| 4 | 0·102 | 148·9 | 132·5 | 111·4 | 107·3 | 106·2 | 100·7 |
| 5 | 0·127 | 133·2 | 118·5 | 99·68 | 96·02 | 95·03 | 90·07 |
| 6 | 0·152 | 121·6 | 108·2 | 90·99 | 87·65 | 86·75 | 82·22 |
| 7 | 0·178 | 112·6 | 100·2 | 84·24 | 81·15 | 80·32 | 76·12 |
| | 0·200 | | 94·46 | | | | |
| 8 | 0·203 | 105·3 | 93·71 | 78·80 | 75·91 | 75·13 | 71·20 |
| 8½ | 0·216 | 102·1 | 90·91 | 76·44 | 73·64 | 72·89 | 69·08 |
| 9 | 0·229 | 99·27 | 88·35 | 74·29 | 71·57 | 70·83 | 67·13 |
| | 0·300 | | 77·12 | | | | |
| 12 | 0·305 | 85·96 | 76·51 | 64·34 | 61·98 | 61·34 | 58·14 |
| 15 | 0·381 | 76·89 | 68·44 | 57·55 | 55·44 | 54·87 | 52·00 |
| | 0·400 | | 66·79 | | | | |
| 18 | 0·457 | 70·19 | 62·47 | 52·53 | 50·61 | 50·09 | 47·47 |
| | 0·500 | | 59·74 | | | | |
| 21 | 0·533 | 64·99 | 57·84 | 48·64 | 46·85 | 46·37 | 43·95 |
| | 0·600 | | 54·53 | | | | |
| 24 | 0·609 | 60·80 | 54·10 | 45·50 | 43·83 | 43·38 | 41·11 |
| 27 | 0·686 | 57·31 | 51·00 | 42·89 | 41·32 | 40·90 | 38·76 |
| | 0·700 | | 50·49 | | | | |
| 30 | 0·772 | 54·37 | 48·39 | 40·69 | 39·20 | 38·80 | 36·77 |
| | 0·800 | | 47·23 | | | | |
| 33 | 0·838 | 51·84 | 46·14 | 38·80 | 37·38 | 36·99 | 35·06 |
| | 0·900 | | 44·53 | | | | |
| 36 | 0·914 | 49·63 | 44·17 | 37·15 | 35·79 | 35·42 | 33·57 |
| 39 | 0·991 | 47·69 | 42·44 | 35·69 | 34·38 | 34·03 | 32·25 |
| | 1·000 | 47·46 | 42·24 | 35·52 | 34·22 | 33·87 | 32·10 |
| 42 | 1·077 | 45·95 | 40·90 | 34·39 | 33·13 | 32·79 | 31·07 |
| | 1·100 | | 40·28 | | | | |
| 45 | 1·143 | 44·39 | 39·51 | 33·23 | 32·00 | 31·68 | 30·02 |
| | 1·200 | | 38·56 | | | | |
| 48 | 1·219 | 42·98 | 38·25 | 32·17 | 30·99 | 30·67 | 29·07 |
| 51 | 1·295 | 41·70 | 37·11 | 31·21 | 30·06 | 29·76 | 28·20 |
| | 1·300 | | 37·05 | | | | |
| 54 | 1·382 | 40·53 | 36·07 | 30·33 | 29·22 | 28·92 | 27·41 |
| | 1·400 | | 35·70 | | | | |
| 57 | 1·448 | 39·45 | 35·11 | 29·52 | 28·44 | 28·15 | 26·68 |
| | 1·500 | | 34·49 | | | | |
| 60 | 1·524 | 38·45 | 34·22 | 28·77 | 27·72 | 27·43 | 26·00 |
| | 1·600 | | 33·39 | | | | |
| 63 | 1·600 | 37·52 | 33·39 | 28·08 | 27·05 | 26·77 | 25·37 |
| 66 | 1·686 | 36·66 | 32·63 | 27·43 | 26·43 | 26·16 | 24·79 |
| | 1·700 | | 32·40 | | | | |
| 69 | 1·753 | 35·85 | 31·91 | 26·83 | 25·85 | 25·58 | 24·25 |
| | 1·800 | | 31·49 | | | | |
| 72 | 1·829 | 35·09 | 31·23 | 26·27 | 25·30 | 25·04 | 23·73 |
| | 2·000 | | 29·87 | | | | |
| 84 | 2·134 | 32·49 | 28·92 | 24·32 | 23·43 | 23·19 | 21·97 |
| | 2·200 | | 28·48 | | | | |
| 96 | 2·438 | 30·39 | 27·05 | 22·75 | 21·91 | 21·69 | 20·55 |
| | 2·500 | | 26·71 | | | | |
| 108 | 2·743 | 28·65 | 25·50 | 21·45 | 20·66 | 20·45 | 19·38 |
| | 3·000 | | 24·39 | | | | |

present, or friction between them and the rim will be insufficient, and there will be relative slip. Here also may be observed one of the factors causing undue wear of the liners and wasted energy.

Another fault of insufficient balls was observed in the very uneven falling caused by interchange of balls among the several layers when the circle was less than a quarter full; but these effects were exaggerated by the smallness of the models used.

When water was used alone, it was observed that 350 revolutions were required to make 2.2 in. of water in an 8½-in. circle continuous, and it fell again on reducing to 217. A greater amount of water could not be made continuous at 400, nor could any amount of mercury alone be made continuous at this speed; 92 revolutions would have been sufficient in the absence of slip. In using water in conjunction with glass beads, I notice that very little lifting of the water can be seen on the ascending side, and it seems probable that in practice the level of water throughout the tube-mill would be very nearly that of the outflow. In a 5-ft. tube-mill this means that about half the fall would be through water. A ball falling through 2 ft. of water from 2 ft. above it would be robbed of most of its pulverizing force, and this brings us to possible improvements, suggested by theory and experimental data.

Linings at present seem to require the presence of the fitter or the mason more than they should; in fact, a duplication of mills seems indicated in the absence of more efficient linings. It seems barely possible that a solution of this question might be had from a continuous outer layer of balls. These would make a sort of automatic lining; any balls worn down would be continually replaced and the layer maintained while the mill is running. A glance at the table will show that the speed of revolution must be kept constant and very nearly free from variation.

Naturally, a movable steel liner would be used behind the balls, but that would only require renewal at much longer intervals than now necessary. A fairly efficient fall would be obtained if the tube were filled about two-thirds with balls. Of course, while relying on fall for crushing power, it is obvious that in all cases it cannot pay to waste energy in splashing water, and the outlet must be arranged so that depth of water does not much exceed depth of balls on the bottom of the tube-mill as they are

in motion. It does not seem certain in wet crushing work that the grinding effect between the rolling balls is not of greater importance, when the fineness of the material and the floating power of the water are borne in mind. If practical experience determines this to be the case, it will only be necessary to drive the mill at such a rate as will establish the angle of fall of the balls in water, which may be somewhere about 30 deg. In this case the tube might be somewhat less than half full, and the outlet could be arranged through a hollow trunnion, as is frequently done. An advantage would be a decrease of wasted energy in lifting balls worn down below a useful size, which seems difficult to obviate entirely at this time.

A practical method of determining the best speed at which to run any tube-mill would be by means of indicator diagrams or measurements of current to determine when the power absorbed, divided by the revolutions per minute, became a maximum.

[In the discussion that followed Mr. White's paper, S. H. Pearce told of a liner that was being tried at Glen Deep, which was made of rings of manganese steel. At first there was a noticeable absence of rumbling. They noticed that the pebbles had a tendency to wear flat, and that the crushing efficiency dropped considerably. Later the rumbling was gradually resumed, and the crushing efficiency increased. His conclusion was that there was considerable slip on the smooth surface when starting, and that later the lining acquired a rough surface and raised the pebbles a little higher, causing crushing instead of reduction by attrition.

W. R. Dowling, of the Robinson Deep mine, said they had two tube-mills running, one with manganese steel and the other with a silex lining. The latter mill took a larger feed and gave a finer ground product than the manganese; the pebbles also retained their rounded shape, while with the steel lining flat faces were in evidence, showing a sliding action.]

# TUBE-MILL NOTES [1]

By Alfred James

(March 16, 1905)

One-stage wet tube-mill work, that is, without return, obtains not only at El Oro, but in Korea, in Germany, and even at Kalgoorlie itself, the Ivanhoe having adopted this method in preference to the return system. The Ivanhoe figures are actually only half those of the other mines, which show their cost per ton milled instead of per ton of sand absolutely slimed. On the other hand, the Ivanhoe has recently thrown out its tube-mill, evidently owing to Mr. Nicholson's preference for pans, and it is possible, therefore, that the cheap tube-mill work accomplished at the Ivanhoe may not compare satisfactorily with the more expensive work accomplished at the Hannan's Star or the Oroya Brownhill. In the latter case the results are complicated by the fact that pans are used.

At first sight, the spitz separation of the coarse particles seems undoubtedly the proper method to pursue, but when it comes to passing 268 tons of sand per diem through the Hannan's Star mill in order to slime the total sand output of 38 tons only, it looks as if steps should be taken to render unnecessary such an enormous amount of return, especially as it is quite possible to slime the material, in one operation only, to a fineness equal to that of the Hannan's Star finished product — less than 5 per cent. retained on a 150-mesh sieve. In this connection it is interesting to note that the costs given by Mr. Grüssner, as well as by Mr. Broadbridge, require to be multiplied by two in order to obtain the actual cost per ton of sand slimed in the tube-mill. Mr. Broadbridge's figures refer to the total sand treated in the agitation plant, but of this over 50 per cent. has been first separated by a spitz and passes to the plant without regrinding, as would be obvious on examination of the grinding analysis.

[1] Abstracted from contribution to discussion, Institution of Mining and Metallurgy, Jan. 19, 1905.

The three division plates introduced into one of the flint-mills at Kalgoorlie proved absolutely unsuccessful in practice. Manufacturers formerly made a blunder in providing divisions in the tube, as it is found that the little flints work their way up the tube through the holes in the dividing plates and refuse to return; they become trapped, and are actually forced out at the feed end of the mill. Now, each mill is but one long tube, and the flints bed themselves perfectly even. There is no overlapping at any particular spot, and no segregation of sizes whatever.

As for the great wear and tear of lining, reported from the Rand, this is due to the deliberate experimenting with coarse feed and increasing output; in regular tube-mill practice there will be only the usual consumption of lining. By "regular tube-mill practice" is meant the proportion of water in feed, rate of revolution, weight and size of flints, and size of machine, for given output. As no one has given figures, with the exception of Mr. Robinson, who gave a maximum figure, it may be added that in grinding sand a pulp of 50 per cent. thickness has been found to give good results, but pulp even thicker (60 per cent.) than this is in successful use, so that there are questions of dilution, before the spitz and amalgamation plates, which have not yet been fully considered. In the experimental work on the Rand the mill men naturally rush their feed through; that is where their experimental work necessarily differs from their practice. The rate of revolution most satisfactory in practice is found to be $200 \div \sqrt{D}$, where D equals diameter in inches. Thus a 4 ft. 1 in. mill should revolve at $200 \div 7 = 29$, the correct number of revolutions per minute. This simple formula was first suggested by Mr. Davidsen.

The correct charge of flints for a mill is found to be $W = 44 \times N$, where W equals the weight of flints required and N the number of cubic feet contained in the cylinder of the tube-mill; but West Australian wet-grinding practice takes nearly 50 per cent. more flints than these figures. As for the size of flints, for wet-crushing, large pebbles are found to be best, say those of from 3 to 4 in. diameter. For dry-crushing, smaller pebbles give a finer product.

In first starting tube-mills there is a tendency to load up with flints. Since experimenting on a large scale is necessarily slow — as can be seen by the time which elapsed at the Hannan's

Star before the present high efficiency was obtained — it is desirable not to increase the load of flints beyond that given by makers without careful investigation. It is a curious fact, which has been noticed all over the world, that it takes much less horsepower to run a tube-mill crushing wet than crushing dry.

One point, which in practice has proved most important, is the need of elasticity in the capacity of tube-mills. In West Australia mills have been put down with the idea of obtaining a unit of an exact size to work with uniform conditions; but as soon as the stamp capacity becomes increased the mills have not the elasticity to cope with the new conditions, and the question arises, in designing new installations, what size of tube-mill would be most satisfactory. It is difficult to lay down a general rule, but there are two guiding principles. The first is that a tube-mill working at its full or normal capacity is doing work under the most even conditions, and, second, that it is practically impossible to increase the output of a tube-mill beyond its normal capacity. In installations where an increased output is being worked it is desirable to put in a tube-mill of greater capacity than at first required, and to revolve this more slowly than the normal rate, or with a less than normal charge of flints. In this way the fineness or coarseness of feed can be regulated to a nicety, whereas the usual recipe of makers for the regulation of the size of the finished product is impracticable with a stamp battery which has only a certain definite output, and through which, therefore, one cannot rush a greater amount of material in order to render the finished product coarser, and containing a minimum proportion of slime. The better method of accomplishing this is that stated above, to diminish the flint charge or rate of revolution.

# TUBE-MILLS

By H. W. Hardinge

(June 8, 1905)

While the tube-mill, or pebble-mill, is not a new device for fine crushing, yet its application to the milling of ore has been so successful as to give it a place among the machines that contribute toward economy and efficiency. Discussions based on experimental facts and direct application are much needed.

The tube-mill is an exception to the greater part of milling machinery in that its capacity usually exceeds the estimate; at least, such is my experience in fine grinding, wet or dry. I have perused most of the articles which have come under my notice, relative to dimensions of tube-mills, the theory of their operation, and the results obtained. I realize that my experience with a tube-mill (that I had constructed within the last year) may practically duplicate data already published; yet the exact conditions under which I am operating may be of interest and use to others. My first experience with the tube-mill, in an experimental way, dates back eight or ten years, at which time I had occasion to grind blast-furnace slag in the dry way, and was then surprised at the results obtained. At the present time I have just completed a 1000-ton wet test of a very tough scale, the composite average mesh of which is given later.

The tube-mills which are now being adopted are generally longer and of less diameter than the one I used for wet grinding, which is 8 ft. long by 6 ft. internal diameter, less the silex or flint-block lining, which reduces the actual internal diameter to about 5 ft. 6 in.; this mill, loaded with 2500 lb. of flint pebbles about 2 in. diameter, and 21 revolutions per minute, receives 6-mesh ball-mill-crushed material and pulverizes it to the sizes shown in the last column given herewith, at a rate of 3 tons per hour, which is really more than the capacity to which the rest of our plant is adapted. In view of the fact that 2500 lb. of pebble

is not more than one-third of the supposed or theoretical load, the real capacity of this mill is undetermined.

| | BALL-MILL PRODUCT | TUBE-MILL PRODUCT |
|---|---|---|
| Through 6-mesh on 20-mesh | 25.9% | 0.0% |
| " 20- " " 40- " | 13.5 | 2.0 |
| " 40- " " 60- " | 9.0 | 8.5 |
| " 60- " " 80- " | 8.0 | 14.0 |
| " 80- " | 40.0 | 75.5 |
| | 96.4 | 100.0 |

A few tons of quartzose ore gave a greater grinding capacity. Tests for consumption of pebbles showed less than 1 lb. per ton of ore treated. The wear upon the lining is also slight, compared with the result obtained. Based upon my present experience, I have placed an order for another tube-mill still further from the general practice, that is, 6 ft. long and 6 ft. in diameter.

Experiments and discussion are needed to determine the most effective diameter and length of mill; revolution under different loads; size of pebbles; whether pebbles should be of uniform or different sizes; what relation size of feed may have to diameter of mill and of pebbles; what is the most efficient feed and discharge; whether the action is percussive-crushing or crushing-grinding, or a combination of both. The capacity and small load of pebbles in the mill I now have in use would suggest a percussive effect; certain it is that much less power to operate is required than when a larger load of pebbles is used, which would lessen the useful duty of the mill, and cause a needless attrition of pebbles and lining.

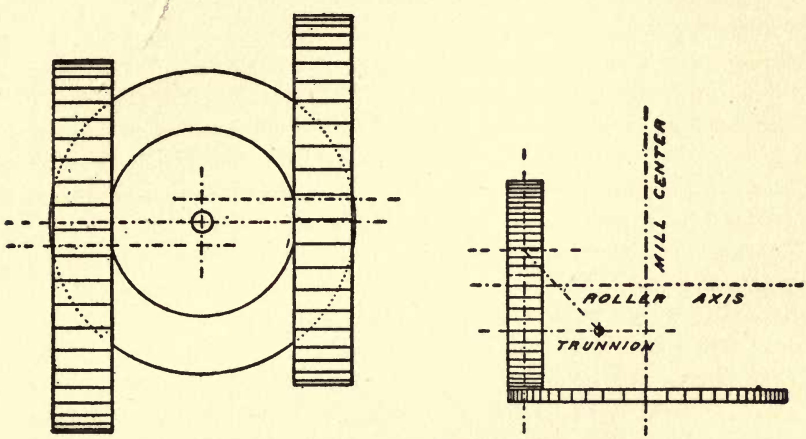
## CHILEAN MILLS

By M. P. Boss

(December 15, 1904)

The Chile mill, as its name implies, was of Latin origin, while the stamp was of Saxon. In its crudest form it was a circular stone with a hole through the center, through which passed a pole, one end of which was attached to a post, while the other end was propelled by an animal. This machine was developed in its native environment into a mill with two wheels, having iron tires and driven by water or steam. Under foreign influence it developed numerous modifications, among which are the fast-motion edge-running roller-mills now quite common. In its original form, with slow motion, the centrifugal action or the tendency to go in a straight line, instead of its circular course, was not great, but when it developed into massive form or into high speed, this tendency became important, and different means were devised to neutralize the outward thrust on the axle. In the massive slow-motion mills, like those of El Bote of Zacatecas, or La Union of Pachuca, the Mantey offset is effectively used. It is also used on some fast-motion mills. Some have rollers inclined toward the center, like a railroad train rounding a curve; others are trunnioned at a point below the axle, so that a portion of the outward thrust is diverted to augment the downward pressure of the roller upon the die. In this day a fast-motion roller-mill that employs no means to utilize, at least partially, the centrifugal thrust of the roller to the crushing can hardly be rated as high-class mechanism because of its wastage of force. The Mantey offset has the axle of the wheel set behind a diameter line to which it is parallel, so that in the turning of the mill the roller, not being true with the die, is more or less shoved over it, while at the same time it revolves. When the offset is properly proportioned to the speed at which the mill runs it is, as before stated, thoroughly successful, but as the resistance is that of

grinding, it has the objection of consumption of power and metal common to grinders in the ratio of this action. When a roller is inclined properly to the speed at which it is run the thrust force is entirely exerted upon the inclined bed or die without loss. A vertical roller, trunnioned at a point below its axle, as in Fig. 32, utilizes the centrifugal thrust of the total weight of the roller, less that portion that lies below a horizontal line through the trunnion, plus its balancing equivalent above said line. With the advent of faster-motion mills, it has become quite general to




FIGS. 31 and 32. — Chilean Mill.

use more than the two wheels or rollers common to Spanish-American mills. If a given weight is put into two rollers, instead of three or more, it is simpler construction, having fewer parts and is presumably cheaper, and it is also more convenient for examination and repairs. A large roller presents a more acute angle to its bed than a small one, for which reason it is less gouged by hard particles in its path, consequently its proportionate wear is less. In the accompanying sketches, Fig. 31 is a plan of mill, showing Mantey offset; Fig. 32 is an elevation, showing the roller trunnioned below its axis.

# PART IV

# DRIERS AND DRYING

## ORE DRYING

By W. R. Ingalls

(From the *Pacific Coast Miner*, July 11, 1903)

In many metallurgical processes, a preliminary drying of the ore to be treated is necessary; especially is this the case in the treatment of ores by processes wherein a fine crushing is required, since not only is it impossible to sift damp ore through fine screens, but also the capacity of the screens is greatly increased by giving them perfectly dry ore at rather an elevated temperature, say 250 deg. F. This is particularly important if the ore be of a clayey nature; ores of such character, even if quite dry, lie dead on the screens and have a tendency to clog the latter if the sifting be done cold. When hot, however, such ores are quite lively and sift as well as hard, gritty ores. Philip Argall, who has given much attention to this subject, illustrates the point by citing the action of wood ashes; if these are poured ever so gently on dry ground when quite hot, they will spread out almost like water, but if cold, they will not spread to any great extent. Although the subject of ore drying is an important one, it has never been treated upon to any considerable extent, so far as I am aware.

If the ore has to be dried, an efficient type of apparatus should be selected. Probably there is nothing better than the well-constructed revolving cylinder drier commonly in use, unless it be a modern zigzag tower drier; if either be used, proper precautions should be taken to avoid loss of fine ore in the form of dust carried off by the chimney draught; this implies the installation of a dust-settling chamber between the cylinder, or tower, and the chimney. The more valuable the ore under treatment the more important is attention to this point.

The simplest form of drier is a series of cast-iron plates placed over a flue so that they will be heated from below by the hot gases from the fireplace, each plate having a flange at its sides through which adjoining plates can be bolted together if desired, by which method fine ore is prevented from sifting through the

joints between the plates. This form of drier is easily adaptable to a utilization of waste heat from other operations. It has the drawback that, if the plates be set horizontally, hand labor is required in spreading and moving the ore, but if the plates be set at an angle the drier can be arranged so that the dry ore will slide off into a conveyor alongside, or directly to the next operation. In drying ore by indirect heat, as in this manner, the capacity of the apparatus is determined largely by the area of the heating surface. In some experiments in drying concentrated zinc blende, which had been sifted through a standard 8-mesh sieve, having apertures of $\frac{1}{16}$ inch, or 1.59 mm., the ore containing 5 per cent. moisture, I found that when spread on an iron plate in a layer one inch thick, at a temperature below the ignition point of the ore, but above the melting point of lead, it could be dried thoroughly at the rate of about 25 lb. per square foot of surface per hour. With moderate stirring it was possible to dry at the rate of 37.5 lb. per square foot per hour, while with a well-designed mechanical drier an efficiency of 50 lb. per square foot per hour would probably be realized. In this case the ore tested weighed 125 lb. per cubic foot, and capacity should be stated in terms of volume rather than in terms of weight.

The zigzag, or gravity, drier is a rational and efficient device, but a rather high belt elevator must generally be installed in connection with it. This form of drier has been highly developed in the Edison tower, which is built in the form of a shaft, 3x3 ft. or more in area and 24 ft. and upward in hight. These driers, which are employed at the magnetic separating works of the New Jersey Zinc Company, at Franklin Furnace, N. J., occupy small floor space at the expense of hight, but in mill buildings the latter is generally cheaper than the former. At Franklin Furnace, N. J., an Edison tower 3x3 ft. and 24 ft. in hight dries ore of 1.5 in. and downward in diameter, containing 4 to 6 per cent. moisture, at the rate of 500 tons per 24 hours, with consumption of 0.12 tons of fuel per hour, the fuel consisting of one-third bituminous coal and two-thirds anthracite of buckwheat size. However, the product from the drier still contains about 1 per cent. of moisture. In other industries, towers of 8x8 ft. in area and 50 ft. in hight are used.

Similar in principle to the zigzag driers are the gravity driers, in which the ore is heated as it slides down a long incline.

Another type of drier which deserves attention is a series of troughs heated from below, or above, or both, through which the ore is caused to travel by means of endless screws, the latter design being, in fact, an adaptation of the ordinary screw-conveyor. Although the wear and tear on such apparatus is rather great, it is at its best when handling fine ore, and if the various parts are properly designed, the cost of repairs and renewals should not be excessive. The movement of the ore through a trough-conveyor can be effected by traveling rakes in the same manner as in a mechanical roasting furnace or by the push plates of the ordinary scraper or drag conveyor. Besides the cost of repairs and renewals, the drawbacks to making driers out of trough conveyors are the comparatively large expense for installing the amount of heating surface that must be provided; and the large consumption of power required in their operation. Driers of this type have been discarded, in some cases, because they have failed to deliver a thoroughly dry product, which means, of course, that a sufficient surface of the ore to be dried was not exposed under the temperature of operation. The increasing use of magnetic separators, which must be provided with perfectly dry ore, has caused the question of driers to become rather an important one. A mechanical development of the ordinary plate drier, heated from below, consists in the installation of a circular hearth of cast iron, 12 to 16 ft. in diameter, on which the ore is fed near the center, whence it is worked peripherally outward by means of plows on a rapidly revolving arm, the dry ore being finally discharged over the edge of the hearth. This arrangement is much cheaper in first cost and, also, in subsequent operation than any trough drier I have ever seen.

The ordinary cylindrical drier is too well known to require special description. The cylinder is commonly made of steel plate, which may be used with or without a brick lining, or of cast iron. The loss in dust is likely to be no unimportant matter in drying fine ore in a revolving cylinder, and the more the capacity of the latter be increased by faster driving and the harder the firing (increasing the velocity of the combustion gases through the cylinder) the greater will be the loss in dust. Hence, a cylindrical drier should be of large size, partly for the sake of exposing a greater surface of ore to the gas, thus permitting slower speed,

and partly to reduce the velocity of the gas passing through the cylinder. Both the Rothwell drier, which consists of a cylinder divided into four quadrants by longitudinal diaphragms, and the Argall drier, which consists of four small cylinders, united by surrounding rings, are logical improvements over the simple cylindrical drier. The Argall drier is made in two sizes, one of which has four tubes, each of 20 in. diameter inside the lining and 17 ft. in length, and the other four tubes, each of 25 in. diameter inside the lining and 25 ft. in length. The former has a capacity of 80 to 100 tons of quartzose ore per day, the latter a capacity of 150 to 200 tons per day. In his paper on "Sampling and Dry Crushing in Colorado," Mr. Argall gives some data as to the efficiency of these driers. At the cyanide mill at Leadville, Colo., a No. 1 four-tube drier, with inclination of 0.75 in. to the foot, making two revolutions per minute, dried 70 tons per 24 hours of soft clayey ore, containing considerable talc and averaging 10 per cent. moisture, down to 1 per cent. moisture, with a consumption of one ton of coal of fairly good quality. At the Bessie mill, at Telluride, Colo., a No. 2 drier treated 177 tons per 24 hours of clayey ore, with consumption of 2.66 tons of coal, the ore containing 8.06 per cent. water before drying and 1.22 per cent. after drying. At the Leadville mill one pound of coal of fair grade evaporated 6.3 lb. of water; at the Bessie mill 1 lb. of poor coal evaporated 4.54 lb. of water; at the mill of the Metallic Extraction Company at Cyanide, Colo., 1 lb. of good coal evaporated 9 lb. of water. These figures indicate a reasonably high utilization of the calorific power of the coals, comparing favorably with the percentage of heat utilization in ordinary steam boilers.

## NOTES ON ORE AND COAL DRYING [1]

BY C. O. BARTLETT

(October 17 and 24, 1903)

Drying by direct heat, as in the rotary cylinder driers, is usually the cheapest method. Great care should be taken in the construction and erection of all direct-heat driers. All iron parts should be designed to allow for expansion and contraction, and all settings and bearings should be very substantial. The steel sheets of the cylinder should run the entire length, and all seams should be longitudinal. There should be no cross seams at all, since they are liable to break.

The cost of drying minerals depends upon five factors, as follows: (1) The percentage of moisture content; (2) upon whether or not the mineral to be dried will permit of the passage of the fire-gases through it without injury; (3) upon whether or not the mineral be sandy or clayey; (4) upon the ignition temperature of the mineral; (5) the degree of dryness to be attained. It is generally safe to estimate on evaporating 10 lb. of water per pound of coal burned in drying ores through which the products of the fire can be passed. Most ores can be dried by direct passage of the products of combustion over them, but some fine clays, and even some kinds of glass sand, will not permit of this on account of danger of discoloration. Silicious minerals can be dried a good deal more easily than clayey ones. In drying coal or other material that has a low ignition point, the temperature must be kept low. There is no danger, however, of burning any material containing a considerable percentage of water, and in drying inflammable materials it is sometimes advisable to use two driers in series, firing heavy on the first one while there is plenty of moisture, and finishing on the second with light firing. It is very much harder to dry ore down to a moisture content of

[1] From a paper read at the American Mining Congress at Deadwood, S. D.

0.5 per cent. than to 2 per cent., and, generally speaking, it is unnecessary to go below the latter figure.[1]

The very successful application of coal-dust firing to the burning of cement in rotary kilns and the extensive use that this system is now finding in the American cement industry direct attention to the means for pulverizing the coal to the required degree of fineness. In order to pulverize coal economically and satisfactorily, it should not contain more than 1 per cent. moisture. The pulverizing capacity of a mill is nearly twice greater with coal containing only 1 per cent. moisture than with coal containing 2 per cent. The moisture content must be expelled from the coal without causing the coal to lose any of its volatile combustible. Two lots of coal will rarely dry alike, some coals giving up their moisture easily and freely, and others with difficulty. It appears that coals in which the ash is composed largely of silica dry easily and thoroughly, while those of which the ash is high in lime or clay are difficult to dry. It is very important to handle the coal in such way that warm air in large quantity be brought in contact with every particle of it, which is best accomplished by passing the current of air from the dried material through that which is wet. It is never safe to pass the fire-gases through the drying coal. The ignition temperature of coals is variable, as is also the temperature at which they will give off their volatile combustible. In general, coal can be safely delivered from the drier at about 150 deg. F. without loss of gas. At 225 deg. F. there is likely to be a small loss of gas, and that temperature cannot be recommended as good practice. It is necessary to use a fan blast to produce a sufficient current of air to carry off the moisture. This will carry off 3 to 5 per cent. of coal dust, which should be saved by passing the current into a brick-dust settling chamber, the walls of which will retain sufficient heat to prevent the moisture from condensing.

[1] This is to be considered rather a high figure. Of course the ratio will vary according to the character of the coal. — EDITOR.

## GRINDING MACHINES USED AT KALGOORLIE [1]

By W. E. Simpson

(November 14, 1903)

In the Kalgoorlie district two kinds of dry-crushers are used, namely, ball-mills and Griffin mills. For wet grinding, flint mills and Huntington mills are employed. Ball-mills have for years given every satisfaction at the Associated, Kalgurli, Hannan's Star and Boulder Main Reef mines, while the Griffin mills claim supporters at the Perseverance, South Kalgurli and Great Boulder Proprietary.

The Griffin mill is exceedingly neat and compact, and occupies only about one-third of the floor space of a ball-mill, but this is of small moment where economy, efficiency, and profit are the chief matters to be considered.

In comparing the output of a Griffin mill with that of a ball-mill, it must be recollected that the former is fed with pieces no bigger than walnuts, while the latter will take lumps as big as the fist.

It was found on one occasion that 75 per cent. of the ore fed to the ball-mills at the Boulder Main Reef was too large to pass through a 2-in. ring; indeed, 3 per cent. proved too large to pass through a 6-in. ring. In spite of this, the mills were each treating 33 tons a day.

A defect of the Griffin mill is its very small screen-area. The pulverized ore is drawn through the meshes by the suction of a Sturtevant fan and is caught by cyclone arresters. It is exceedingly complicated and easily gets out of order, while it possesses the additional disadvantage that, as the pendulum must revolve at an enormous speed, the wear and tear is very great. Of the 12 Griffin mills installed on the Great Boulder Proprietary mine,

[1] Abstract from paper entitled "Treatment of Telluride Ores by Dry-Crushing and Roasting at Kalgoorlie, Western Australia," Institution of Mining and Metallurgy, October, 1903.

never more than 10 are running at one time; the others are always under repair. The wearing parts, i.e., the roll-tires and followers, costing $15.60 and $10.08 each, respectively, require renewing every eight days, while the die rings, costing $19.20 each, and the roll bodies, costing $15.12 each, last only about six or eight weeks. On the South Kalgurli the pendulum shafts are replaced on an average every two months, at a cost of about $48 for material alone. Driving-belts also suffer severely, and require to be renewed twice a year. The power required is 20 indicated horse-power per mill, and the output is 1.25 tons per hour of running.

It is now admitted generally that among the dry crushers the ball-mill has no rival as regards output, low running cost, and the excellence of its finished product for the subsequent roasting treatment. The screen-area is enormous, and amply sufficient to secure the discharge of the ore immediately it has been reduced to the required degree of fineness, and as the screens themselves revolve they are subjected to a gentle internal scouring, which keeps them clean and in proper trim for efficient working. The cost of maintenance is exceedingly moderate, one steel ball, weighing 18 lb., being put into each mill every day with the charge of ore in order to make up for the consumption of metal. The other wearing parts are the grinding plates and side-liners, which are replaced in complete sets every seven or eight months, and cost $864 delivered on the mine. The driving-belts on the Boulder Main Reef, which are of camel's-hair, have now been in use continuously for over four and a half years, and look as if they would still last a long time.

Each mill makes exactly 25 revolutions per minute, requires about 24 i. h. p., and has an output of over 1½ tons per hour. The total weight of the steel balls in each mill varies from 2240 lb. to 2464 pounds.

An examination of the powdered ore leaving the ball-mills on the Boulder Main Reef, where a screen of 20 holes to the linear inch is used, furnished interesting data. A sample assaying 16 dwt. per ton was treated on a series of brass-wire sieves. The portion retained on each was carefully weighed and assayed separately, with the following results:

| | | | | | | oz. | dwt. | gr. |
|---|---|---|---|---|---|---|---|---|
| | Retained on | 40-mesh sieve | | 24.1% | assay. | 0 | 7 | 20 |
| Passing 40 | " | " 60 | " | " | 9.4.......... | 0 | 11 | 18 |
| " 60 | " | " 80 | " | " | 6.2.......... | 0 | 13 | 7 |
| " 80 | " | " 100 | " | " | 6.6.......... | 0 | 17 | 0 |
| " 100 | " | " 150 | " | " | 3.2.......... | 0 | 18 | 7 |
| " 150 | " | " 200 | " | " | 4.2.......... | 0 | 18 | 7 |
| " 200 | " | " ... | " | " | 46.3.......... | 1 | 0 | 6 |

It will thus be seen that as regards value the ball-mills produce only two qualities; the coarse, largely made up of hard particles of quartz, is comparatively poor, while the fine contains most of the gold, owing to the friability of the sulphides and tellurides.

The Griffin mills, even with a coarser discharge screen, yield a much finer product. For example, the Griffin mill at the Great Boulder Proprietary mine, with a 15- to 18-mesh, produces an impalpable powder, of which 75 per cent. will pass through a 150-sieve. It is claimed that the extra work thrown upon the rock-breakers in preparing for the feed of the Griffin mill is compensated by a corresponding diminution of work in the Wheeler pans in the sliming section of the process; while this is admitted, to a certain extent, it cannot be denied that the very fine crushing is a distinct disadvantage for the roasting. A certain amount of grit is always advantageous, as it allows the oxidizing atmosphere to play more effectively in and about the fine sulphurous particles; while at the same time it prevents that banking into ridges, or packing, so noticeable in the treatment of the more finely grained or floury material. Actual practice shows that one 5-ft. Wheeler pan will deal with a daily output of 16.6 tons from the Griffin mills and only 12 tons from the ball-mills; but the consumption of pan shoes and dies is precisely the same, being 1.8 lb. per ton of original ore in either case.

One other point in favor of the ball-mill is that when, as not infrequently happens, a dynamite cartridge is carelessly shoveled up in the mine and sent up with the ore, its explosion does little or no damage to the ball-mill because of the great space available in which to expend its energy. This is not the case with the compact Griffin mill, where the explosion is confined and often blows the bottom to pieces. The expense incurred in repairing the results of an accident of this kind amounted formerly to $192 to $336; but these figures are now considerably reduced through

the substitution of false bottoms of sheet iron for the heavy castings. However, even now the cost of repairs is heavy, besides which, time is lost and the mill is lying idle while the damage is being put right.

Another type of pulverizer, much favored on this field, is the flint mill. It is a hollow revolving steel cylinder, 16 to 20 ft. in length and 4 or 5 ft. in diameter, and closed at both ends except for a small central opening in each. It is lined completely with cast-iron or steel plates ¾ to 1¼ in. thick, and when in operation is partially filled with a charge of hard flint nodules. The sand is forced from the nozzle of a spitzkasten through the central opening in the receiving end, is ground in the mill, and escapes as slime at the discharge orifice. One flint mill is sufficient for a 2000-ton per month plant, using a 30-mesh wire screen, and the work is done so efficiently that 99 per cent., or practically the whole, of the finished product will pass through a sieve of 200 holes to the linear inch.

A complete set of liners weighs about 4 tons, lasts from 7 to 10 months, and costs $384 on the spot.

The working charge of flints ranges from 4 to 6 tons, and about 1 cwt. of flints is consumed in grinding 100 tons fine enough to pass through a wire screen with 30 holes per linear inch. Each mill consumes from 18 horse-power to 35 horse-power, according to size, makes 30 revolutions per minute, and, when working efficiently, emits a low, dull note or roar easily recognized by an experienced attendant.

The function of a flint mill is to pulverize, and as such it is at present unsurpassed, while the pan (which is also used in the treatment at Kalgoorlie) has not only to grind, but also to amalgamate.

Huntington mills have been tried for pulverizing the roasted ore, but the ever-present plaster-of-paris soon encrusts and enamels all the exposed ironwork, the fine mill screens rapidly become choked, and further work is impossible.

# PART V

# CONVEYORS AND ELEVATORS

## MECHANICAL CONVEYORS

By W. R. Ingalls

(April 21, 1904)

Mechanical conveyors, of which there is a great variety, may be classified as of (1) the push or drag type, and (2) the carrying type. In the former, the material is pushed or dragged forward in a trough. In the latter type, it is continuously carried forward on a belt, or in a series of connected pans or buckets, which take the place of a belt. In a horizontal conveyor the only mechanical work to be done consists in the overcoming of friction. It is obvious, therefore, that a well-mounted belt or series of buckets can be moved with less friction and therefore requires less power than any form of conveyor in which the material has to be pushed or dragged forward.

All of these conveyors are used in practice, some of them extensively. Some of them are extremely efficient machines; others have very little to commend, yet are useful for some special purposes because of limitations in the application of better types. The special form of conveyor must always be chosen with view to the work that is to be done. In this article, I have reference only to the use of conveyors for the transportation of ore and other mineral substances. There is a dearth of practical information on this subject; even the manufacturers appear to lack a good deal of important data. It is obviously a subject in which experiences may differ widely under varying conditions.

### Push or Drag Conveyors

Among the conveyors of this type are the screw, the scraper, and the reciprocating. All of them have the advantage that material can be discharged, without complicated machinery, at any desired point, which makes them especially useful for the filling of a series of bins.

*Screw-Conveyor.* — The screw-conveyor is one of the oldest of conveying devices. Also, it is perhaps one of the most inferior. The screw-conveyor consists commonly of a trough of iron or steel, with semi-cylindrical bottom, in which is turned an endless screw, composed of a shaft, solid or hollow, and a spiral of steel or cast iron. The shaft is supported in boxes at each end of the trough, and by intermediate hangers in long conveyors, and is driven by pulley, gear, or sprocket wheel. The shaft is generally made in sections, which may be united in any suitable manner, though certain devices are much better than others. The spiral is ordinarily of 8-in., 10-in., or 12-in. diameter. In transporting ore it is inadvisable to turn a 9-in. or 10-in. screw at more than 50 to 75 revolutions per minute, since a higher speed is apt to throw material out of the trough and produce too much dust. Obviously, the speed should diminish as the diameter of the screw increases.

The capacity of a screw-conveyor depends upon the diameter and pitch of the screw, its speed of revolution, and the specific gravity of the material to be transported. One manufacturer gives the capacity of a 6-in. screw, run at 100 revolutions per minute, at 3 tons per hour; of a 9-in. screw at 70 revolutions per minute, 8 tons per hour; and of a 12-in. screw at 50 revolutions, 15 tons per hour. It is presumable that these figures for capacity refer to quartzose ore, which may be taken as weighing 100 lb. per cu. ft. Another manufacturer estimates the capacity of a $5\frac{7}{8}$-in. screw at 120 revolutions, 42 cu. ft. per hour; $7\frac{7}{8}$-in. at 110 revolutions, 71 cu. ft.; $9\frac{7}{8}$-in., at 100 revolutions, 141 cu. ft.; $11\frac{3}{4}$-in., at 80 revolutions, 247 cu. ft. It is quite right to state these data in cubic feet, instead of by weight, but the speeds given are too high for good practice. However, the capacities appear to be stated moderately, notwithstanding. On the basis of material weighing 100 lb. per cu. ft., the capacity of the $5\frac{7}{8}$-in. screw would be 2.1 tons per hour; of the $7\frac{7}{8}$-in. screw, 3.55 tons; of the $9\frac{7}{8}$-in. screw, 7.05 tons; and of the $11\frac{3}{4}$-in. screw, 12.35 tons. The figures of either of these manufacturers seem to be on the safe side as to capacity, since a 9-in. conveyor run at 70 revolutions per minute will certainly transport 10 tons per hour of ore weighing 150 lb. per cu. ft., or $6\frac{2}{3}$ tons of ore weighing 100 lb. per cu. ft.

Ideas as to the power required to operate a screw-conveyor

are less definite. In the transportation of any substance horizontally, friction is the only element which has to be overcome, not only the friction of the material itself, but also that of the mechanism. It is evident, therefore, that the power required is a function of the weight of the material, the distance to which to be carried and the speed, plus the similar factors for the mechanism. One manufacturer states that a $5\frac{7}{8}$-in. screw run at 120 revolutions per minute requires 0.5 horse-power per 33 ft. of length; a $7\frac{7}{8}$-in. screw at 110 revolutions, 6.75 horse-power; and a $9\frac{7}{8}$-in. screw at 100 revolutions, 1 horse-power. These figures are rather lower than practice indicates, and would appear to correspond more closely to the power required to drive the conveyor empty than full. Another manufacturer gives the formula, H. P. $= WL \div 3 \times 33{,}000$, in which W is the weight in pounds of the material to be carried per minute, and L the distance in feet to which it is to be carried. According to this, the power required to carry 10 tons of ore 100 ft. per hour would be only 0.33 horse-power, which of course is absurd, since it would require far more power than that to run the conveyor empty. A 9-in. screw conveying that quantity of material would probably require 4 to 5 horse-power. The formula should evidently be expressed as H. P. $=[WL \div (3 \times 33{,}000)] + FL$, in which F stands for the power required to turn the screw itself at a specified speed. The screw is wasteful of power, because not only is the ore pushed through the trough as in the scraper-conveyor, but also the screw presents a greatly increased frictional surface, while it is subject to all the frictional resistance of a poorly supported and carelessly attended line of shafting, running in grit all the time.

The screw-conveyor is the cheapest of all conveyors to install. A 9-in. screw, 100 ft. long, ought to be put up for about $300. On the other hand, all of its parts are subject to heavy wear, and repairs and renewals may easily amount to 100 per cent. per annum, this depending upon the work required of it. There are some cases wherein it is advantageous to use a screw, notwithstanding its serious drawbacks. They are at their best when used for finely crushed and dry ore. They are more troublesome with wet, clayey ores, and are quite unsuitable for coarse ores. A very long screw is apt to be a nuisance anyway. A short screw often makes a good feeding device. The screw-conveyor

with externally heated trough has been proposed as a drying and roasting furnace. It has been used occasionally for the former purpose, but not for the latter. Neither arrangement commends itself.

*Rotary Conveyor.* — The screw-conveyor is often referred to as a spiral conveyor. Another form of spiral conveyor consists of a cylinder with an interior spiral, the cylinder being supported on rollers and revolving like a cylindrical roasting furnace. Conveyors of this form are seldom used. They would appear to be costly, clumsy, and difficult to repair, while material can only be fed at one end and discharged at the other end, which in adaptability would make it the least advantageous of all conveyors.[1] If the cylinder be set on an incline, or if it have a taper, of course no interior spiral is necessary. The cylindrical drier and several forms of roasting furnaces are really forms of this type of conveyor, just as other mechanical drying and roasting furnaces embody the principle of the scraper-conveyor. Roasting cylinders as long as 60 ft. are used in Europe, and cement kilns as long as 120 ft. are used in the United States.

*Scraper Conveyor.* — The scraper-conveyor consists essentially of a trough in which the ore is dragged forward by a series of transverse push-plates called flights. The method of connecting the push-plates is subject to a large number of modifications. Thus there is the continuous cable, dragging circular flights through a V-shaped or semi-cylindrical trough, and the monobar conveyor, in which the flights are carried by a series of single linked bars. One of the commonest forms of this type of conveyor is, however, the double link-belt chain, supported on rollers, wheels or sliding shoes, which run on rails at each side of the

[1] Morris M. Green, in the *Engineering and Mining Journal,* May 26, 1904, criticized the statement that "material can only be fed at one end," saying that he knew of one conveyor of this sort which has five inlets, receiving material discharged from as many furnaces. This type of conveyor may be costly, but such cost might be justified by advantages, under certain circumstances. As a cooling conveyor, it is remarkably successful where materials like cement clinker are to be conveyed and cooled simultaneously, without loss of objectionable dust into the atmosphere of a mill, where it can embarrass workmen. Material in a rotary conveyor can be cooled by drawing air through the conveyor tube, and the hot air can be utilized. Also water can be sprinkled on the conveyor's exterior, where the nature of the material does not allow direct contact with water.

trough, carrying the flights between them. This is known as the suspended-flight conveyor. The chains pass over sprockets at each end of the conveyor and return on overhead rails. The sprockets at one end are keyed on the driving-shaft, while those at the other end are carried in boxes which can be adjusted to take up the slack in the chains. The monobar conveyor can be constructed so as to make a bend in the horizontal plane, or even make the complete return circuit.

The scraper-conveyors have the advantage that they can be arranged to be fed or to discharge at any point. They have the disadvantages of involving a good many wearing parts and requiring considerable power to drive. The Link-Belt Engineering Company gives the following formula for power:

$$\text{H. P.} = (\text{ATL} + \text{BWS}) \div 1000,$$

in which A and B are constants depending on angle of inclination from the horizontal, T is the tons per hour to be conveyed, L the length of the conveyor in feet, center to center, W the weight in pounds of chains, flights, and shoes, and S the speed in feet per minute. For horizontal runs, $A = 0.343$ and $B = 0.01$. According to this formula, the power required to move 10 tons of ore per hour the distance of 100 ft. would be 3.5 horse-power, but I should hesitate to reckon so low. Anyway, it always requires more power to start a conveyor than to operate it, and therefore a larger motor should be provided. Scraper-conveyors are usually operated at speeds of about 100 ft. per minute. The weight of the chains, scrapers, wheels and axles, or rollers, amounts to about 30 to 35 lb. per foot, center to center, for a 10-in. or 12-in. suspended-flight conveyor, which at 100 ft. travel per minute will have capacity for moving about 10 tons per hour of ore weighing 150 lb. per cu. ft. The cost of a suspended-flight conveyor 100 ft. long, installed, will come to about $450.

The capacity of a scraper-conveyor depends upon the width of the trough, the speed of the chain, the volume of the ore, and the frequency of the flights. The flights are commonly set 16 in., 18 in., or 24 in. apart. Obviously, the flights will not push the ore ahead in an even sheet, but will crowd it up into little heaps, a succession of which will be moving through the trough. Therefore, the more frequent the flights, the greater the capacity of the conveyor. The suspended-flight conveyor is superior to

other forms; it requires about 20 per cent. less power than the simple drag, runs more smoothly and is not so noisy. The point of special weakness in these conveyors is the chains, the breakage of which is likely to cause costly and vexatious delays. The monobar is better than the chains; the latter, if used, should be provided of greater strength than is frequently the case. The scraper-conveyor gives the best results with fine ore and moderate lengths. Many examples of large and long installations for the handling of lump ore, coal and rock are to be seen. They are very noisy and are subject to frequent breakdowns.

*Reciprocating Conveyor.* — The reciprocating conveyor is a new modification of the scraper-conveyor, which is finding considerable favor.[1] In this the ore is pushed forward in a trough by a series of flights which are hinged at regular intervals to a ladder-like frame, composed of a pair of channel beams joined by suitable cross-bars and mounted on rollers. This frame is given a reciprocating motion by a crank mechanism, which can be placed at any convenient point. In another form, the flights are fixed to a reciprocating rod, such as an iron pipe of suitable strength, which is supported by wheels and axles. In either case, the flights are so hinged that in their forward motion they bear against stops, and push the material along, while in the backward motion they return to the starting point by dragging back over the top of the material. In this way the ore is literally shoveled forward, stroke by stroke.

In the gold mills at Kalgoorlie, Western Australia, the reciprocating conveyor has superseded all others, being commended for its simplicity and low cost for repairs. As used there it consists of a semicircular trough about 60 ft. long, provided with a ladder-like frame, with blades hanging from the rungs, which is made to move horizontally to and fro on rollers. The blades are free to swing in one direction, but are prevented by a stop from swinging further back than the perpendicular. When the ladder is traveling forward, each blade hangs vertically, and

[1] A correspondent writing in the *Engineering and Mining Journal*, of May 5, 1904, criticizes the statement that the reciprocating conveyor is a modification of the scraper-conveyor. This is, however, a mere matter of classification. He remarks further that the late Eckley B. Coxe designed and installed a reciprocating conveyor at an anthracite breaker some time in the 80's — about 1886. It was not a success economically, because of the cost of repairs and renewals.

pushes a little heap of ore before it for a distance of 20 in., the length of the stroke. (See Fig. 33.) On the return stroke the blades, being free to swing, slip over the tops of the little heaps, and on the completion of the stroke resume their original vertical position. (See Fig. 34.)

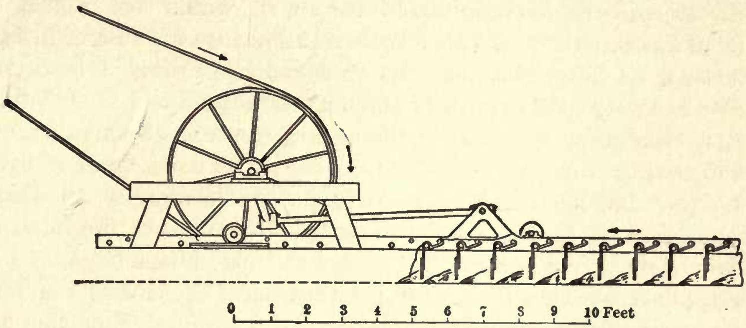



Fig. 33. — Push Conveyor, Forward Stroke.

The reciprocating conveyor has these advantages: It can be fed and discharged at any point; it occupies less hight than the chain scraper-conveyor; and all of its wearing parts, which

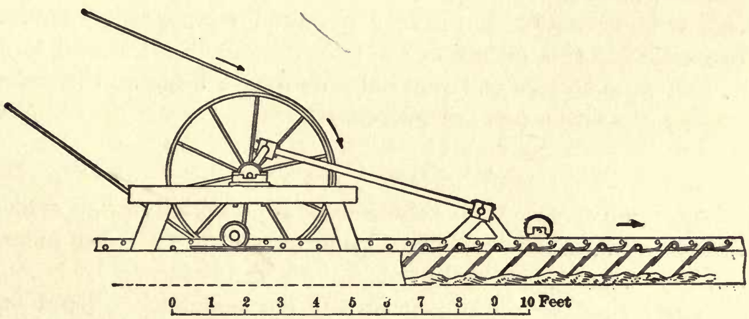



Fig. 34. — Push Conveyor, Backward Stroke.

anyway are comparatively few, are outside of the grit, save the flights themselves and the trough. On the other hand, it is uneconomical of power, owing to the frequency with which motion is reversed. At every stroke the inertia of the entire lot of ore in the trough has to be overcome, and this will probably

limit the usefulness of this type of conveyor to a comparatively moderate length. Moreover, they are obviously inapplicable to conveying materials containing lumps. They are considerably more costly than the ordinary scraper-conveyor, the cost varying according to the details of manufacture. Thus, to install a reciprocating conveyor 100 ft. long, capable of transporting 10 tons per hour of ore weighing 150 lb. per cu. ft., would cost from $700 to $1200 (actual quotations, with an allowance for cost of installation). A 15-horse-power motor should be provided to drive. The capacity of this form of conveyor is determined by substantially the same factors as in the case of the scraper-conveyor.

Another form of reciprocating conveyor consists of a light trough, supported or suspended in a suitable manner, to which a to-and-fro movement is imparted by suitable mechanism. This form of conveyor is not in general use, but I have seen it employed with good success for transports of several hundred feet, the entire installation being of the simplest construction. Obviously, however, it is suitable only for fine, dry material, or else a loose pulp. In either case, the forward travel of the material will depend upon the slope of the trough and the length and number of the jerks. The Wilfley conveyor, which is of this type, is used for the transport of wet concentrates, the motion of the trough being given by the same mechanism that is used for the Wilfley table. A recently patented reciprocating trough-conveyor has the bottom of the trough made in a serrated form, so that at each jerk the material goes over a ledge and therefore attains a positive forward movement.

## Carrying Conveyors

The conveyors of this type consist substantially of an endless belt, or a continuous chain of pans or buckets. There are numerous modifications of both forms.

*Belt Conveyor.* — The belt conveyor is essentially a band supported on idlers and running over pulleys at either end, by one of which it is driven. A suitable arrangement at the other end serves to take up slack and keep the belt tight. The simplest conveyor of this type has a flat belt, which has to be quite wide in order to prevent material from spilling off. To obviate this, the belt is concaved, and to reduce the wear of the belt by being thus flexed it is manufactured in various ways. There is also a great

variety in the composition of rubber employed and in the design of the supporting rollers. Rarely, a flat belt with side rims is run over plain rollers.

Irrespective of these modifications in design and construction, the belt conveyor is for many purposes the most efficient of all conveyors. It requires the least power to drive, save for the highly developed forms of continuous bucket conveyors; its first cost is moderate, and the expense for repairs and renewals is less than for any other form of approximately equal first cost. It is adapted to a great variety of uses, carrying ore up considerable inclines and at changes of angle, and has great capacity. By means of a tripper, which has been greatly improved in design, discharge can be effected from the belt at any desired point. It is possible, moreover, where electric power is available, to install a movable conveyor, run by a self-contained motor, and to cause the belt to discharge over the end into any one of a series of bins, by moving it forward or back; and the direction of the belt travel can be reversed. Thus, a line of bins 200 ft. long can be filled by a conveyor of a little more than half that length, the feed being received midway in the line of the bins. Similarly such self-contained conveyors can be constructed in portable form and used for work about the yard, such as the loading of railway cars. These are things which cannot be done so conveniently with any other type of conveyor. Moreover, this can be used as a sorting belt at the same time as a carrying belt, and in taking ore to breakers and rolls a magnet can be set over the belt to pick out drill points and other undesirable pieces of steel and iron.

The rubber belt is quite durable and it may be reinforced on the wearing side by an extra layer of rubber, like elevator belts. It is, however, unsuitable for carrying ore from driers, etc., which is of such temperature as to affect the rubber. The limit of rubber belting in this respect is soon reached (it would be unsafe to attempt to carry ore so hot as 150 deg. C.), but in such cases the Leviathan or Gandy belts may be substituted. Such cotton-duck belts are, however, less durable against abrasion than the rubber.

The capacity of a belt conveyor depends upon the width and speed of the belt and the weight of the material to be carried. If the belt is troughed it is safe to estimate that the load will cover one-half of the total width of the belt, and that the depth

in the center will be one-quarter of its own width. The cross-sectional area of the load (which may be considered as an inverted triangle) multiplied by 12 will give the number of cubic inches of material per running foot of length, and from the weight of the material and speed of the belt the capacity may easily be calculated, but an allowance must be made for irregularity in feeding. A flat belt will carry only about one-third as much as a troughed one.[1]

A belt speed of about 300 ft. per minute is commonly used, but 450 ft. per minute is not excessive; belts have been observed to run smoothly at a speed as high as 900 ft. per minute, but the wear on both the belt and the idlers was then excessive, and so high a speed in no way is to be recommended.

A troughed 12-in. belt, run at 100 ft. per minute, is able to carry 187.5 cu. ft. per hour, or 14 tons of ore weighing 150 lb. per cu. ft., which would be ample for the duty that I have assumed for other conveyors in this article, viz., the transport of 10 tons per hour. The cost of such a conveyor installed would be about $600 for a length of 100 ft. It would require less than one horse-power to drive, assuming it to be properly installed. No general rule can be given for estimating the power required to drive a belt conveyor, which depends largely on the arrangement of the idlers. If they are too far apart the belt will sag down between them, increasing the load; if they are too near together the frictional resistance is increased. The greatest item of repairs in connection with a belt conveyor is the replacement of the belt, which is the most costly single piece of the apparatus. If the belt lasts five years the cost of repairs will come to about 12.5 per cent. per annum; a belt life of only 2.5 years would mean a repair cost of about 20 per cent. per annum. In a certain large works where a good many belt conveyors are employed the actual expense for repairs is not much more than 12.5 per cent. per annum.

It is to be remarked that the belt conveyor is a type of great capacity, and for the transportation of large quantities of material, for which it is especially adapted, it appears much more favorably as regards first cost, operating cost and maintenance than for the transportation of the relatively small quantity of

[1] Thomas Robins, Jr., *Transactions* American Institute of Mining Engineers, XXVI, pp. 78-97.

material which for purpose of comparison has been assumed in this paper.

*Continuous Bucket Conveyors.* — The pan and bucket conveyors consist essentially of an endless chain of overlapping pans and buckets, which may be arranged in a great variety of ways. One of the simplest is the endless traveling-trough conveyor (referred to also as the open-trough conveyor and apron conveyor), consisting of a series of overlapping sections of light sheet-steel trough, which are secured on the under-side to a heavy link-belt chain (or to a pair of chains); the chain passes over a sprocket at each end of the conveyor and the pans are supported on rollers attached to the frame. These conveyors are considerably more expensive than the belt conveyors. The first cost of a 12-in. conveyor of this type, which would have capacity for 10 tons of ore per hour, would be in the neighborhood of $11@$12 per foot, installed. Ordinarily they have the disadvantage of being able to discharge only at the end, where the pans pass over the tail sprocket (although, in the forms wherein the pans are carried between a pair of chains, they can be arranged to dump at intermediate points by having a dip in the rails), and in this respect are of more limited application than the belt conveyors, but on the other hand they are suitable for conveying hot material or substances that would injure a belt. Conveyors of this type, of heavy construction, are used at various places for the transportation of hot slag, and when properly installed give good service. It is only a little step further to the casting and conveying machines for pig iron and other metals.

For the transportation of ore and coal and such substances the highest development in conveyors of this class is to be seen in the Hunt and the McCaslin and similar designs. These consist of a series of deep buckets, which overlap so as to prevent spilling of material between them while being fed, carried on wheels running on rails. The whole contrivance is virtually a chain, or endless train, of small cars. In the Hunt conveyor, the train is moved by an engine having an arrangement of gears and pawls so disposed as to engage the cross-rods or rivets connecting links in the chain to which the buckets are fixed. This gives a continuous pushing action and an even and practically noiseless motion of the conveyor. The buckets may be discharged at any desirable point by adjusting a simple lever which engages with

each bucket and turns it completely over. The buckets may also be fed at any point; this is done usually by means of a mechanical device which insures a feed directly into the buckets. Other manufacturers accomplish the same results in different ways.

These conveyors have the widest range of usefulness. They can be arranged to work at any desired angle and make any desired turn in the same vertical plane. They can transport ore horizontally, then vertically, and then horizontally again, and discharge anywhere on the line. They operate noiselessly and require comparatively little power and very little attention. Repairs and renewals are very low indeed. The capacity is large, depending of course on the size of the buckets and their speed. Unfortunately the first cost is also large. Thus, a conveyor of first-class manufacture, with 18- by 24-in. buckets, running at 10 ft. per minute, which would have a capacity for 10 tons of ore per hour, would cost about $3600 for a 100-ft. length, this cost including the driving mechanism and electric motor. This is much more costly than any other form of conveyor that has been described herein. However, probably only about 1 horsepower would be required to drive it, and repairs and renewals over a considerable period of years ought not to average more than 1 or 2 per cent. per annum on the first cost. Such low figures are actually attained in practice. These conveyors are therefore to be recommended for important installations wherein belt conveyors cannot be used, where the first cost is a minor consideration and the assurance of certainty in operation is an essential.

## BELT ELEVATORS

By W. R. Ingalls

(August 15, 1903)

In almost all ore-milling operations one of the most useful and necessary, and at the same time the most objurgated, pieces of apparatus is the belt elevator. Yet the belt elevator, if properly designed, should not give any more trouble than more complicated machinery, which is exposed to equally hard conditions of service. The design and construction of belt elevators are frequently, however, not what they should be, and indeed authorities differ materially as to the nature of the specifications. Some advocate the perpendicular elevator, others the slanting. Some believe it to be the best practice to feed the ore directly into the buckets, others arrange the feed so that the buckets will scoop the ore from the boot. Some use a special boot, others do not. There are equally important differences of opinion as to the speed of the belt, spacing of the buckets, discharge of the ore from the head end, etc.

The continuous band for an elevator of this type will be of rubber belting, except in the case of the elevator taking the discharge from the drier in mills wherein the ore is dried, if the latter be so hot that a rubber belt would be softened or otherwise deteriorated; in which case link belts or chains passing over sprockets must be used. In drying ore, in order to insure good screening, it has been found to be desirable not merely to drive off the moisture, but also to heat the ore to a temperature of about 250 deg. F., since the hotter the ore the greater is the capacity of the screens, especially if the ore be of a clayey nature. There is, however, a limit to the heating of the ore, which is soon reached if it is to be handled directly from the drier by means of a belt elevator.

The belting for an elevator of this type should be of superior quality, not less than 6-ply in thickness for an 8-in. or 10-in. belt.

Some authorities recommend that a 10-in. belt should be of 7-ply, a 12-in. belt of 9-ply, a 14-in. belt of 9- to 11-ply, and a 16-in. belt of 11- to 13-ply. Such heavy belts as those last mentioned are rarely employed, and their use is of doubtful advisability in connection with the comparatively small head and foot wheels that are commonly provided. However, it appears rational enough to increase the thickness of the belt as the width increases, since the capacity of the buckets and, consequently, the strain on the belt increase at a much greater ratio. An improvement over the ordinary belt is the special elevator belt, which is made by nearly all the manufacturers in this line, it being ordinary belting surfaced on the outer side, or on both sides, with a covering of $\frac{1}{16}$ in. or $\frac{1}{2}$ in. of pure rubber. This covering adds about 12½c. per square foot of surface to the cost of the belt, if $\frac{1}{16}$-in. thick, and twice as much if $\frac{1}{8}$-in. thick. This makes a considerable increase in the first cost of the belt, and opinions differ as to whether the increased wear justifies it.

The buckets commonly employed are the deep form, referred to as "Style A," made of malleable iron. These buckets are seamless, strong, and smooth, and their round corners tend to insure free delivery of the material handled. The buckets are commonly spaced on the belt, from 12 to 20 in. apart, center to center. A spacing of 18 in. is probably the best practice, and when a nearer spacing is employed it is likely to be due to the attempt to obtain too large a capacity with too small a belt. The capacity of an elevator is solely a function of the volume of the ore, the speed of the belt, the spacing of the buckets, and the size of the buckets. The buckets should not be reckoned as running more than one-third full. Assuming that the buckets are running one-third full, are spaced 18 in. apart, and that the belt speed is 300 ft. per minute, the capacity of the sizes of elevator most commonly used, in terms of ore weighing 125 lb. per cubic foot, is as follows:

| WIDTH OF BELT | DIMENSIONS OF BUCKETS | CAPACITY OF EACH BUCKET | CAPACITY OF ELEVATOR IN TONS PER HOUR |
|---|---|---|---|
| 12 in. | 10x6x5 in. | 160 cu. in. | 22.5 |
| 10 in. | 8x5x4 in. | 108 cu. in. | 15.0 |
| 8 in. | 6x4x3.5 in. | 50 cu. in. | 7.0 |

The head wheel should be of sufficient diameter to afford proper friction for the belt. It is seldom advisable to employ a head wheel of less than 30 in. diameter, but it is seldom necessary to exceed 36 in. The diameter of the foot wheel is not of so much importance, but it should not be so small as to cause thereby too sharp a curve in the belt, depending upon the thickness of the latter. A foot wheel 20 in. in diameter is about as small as ought ever to be used, while a 30-in. wheel is probably as large as any elevator will require. The use of a foot wheel 24 in. in diameter is a common practice. The smaller the pulleys the more detrimental to the belt. The belt should always be 2 in. wider than the buckets.

The best speed for a belt elevator is about 300 ft. per minute. In order to impart that speed, the head wheel must run at 38.2 revolutions per minute if 30 in. in diameter and 31.8 revolutions if 36 in. in diameter. With a belt speed of 300 ft. per minute, the centrifugal force throwing the ore out of the buckets will insure a satisfactory discharge, whether the elevator be perpendicular or sloping.

As between perpendicular and sloping elevators, the former generally adapts itself the better to the construction of the building, and permits in many cases a simpler arrangement of the machinery; it is less expensive in first cost and less expensive to keep in repair than the sloping elevator. The drawback to its use is the trouble in maintaining proper tension in the belt, as the latter stretches. This must be arranged for by the installation of a tightener, which is usually provided in the form of an adjustable boot, wherein the foot wheel can be depressed as required. If there be insufficient room for lowering the boot, the head of the elevator may be so arranged as to be raised, but the latter expedient is seldom practised, although it is quite convenient in case the driving belt from the line shaft has a tightener on it, or if the line shaft and the head of the elevator are at the same height and not too close together. In the case of the sloping elevator, inclined at 10 deg. to 15 deg. from the vertical, or more, the belt is always self-tightening, with an elastic tension, the sag on the descending side maintaining a sufficient tension, although the belt may have stretched to a considerable extent. In one case, a sloping elevator, with 10- by 6-in. buckets and a new belt, ran continuously for nine months without any shortening of the belt being required.

There is equally important difference of opinion as to the best manner of feeding the ore into the elevator. Some mill-men advocate that the chute leading into the boot should deliver the ore 16 to 18 in. above the center line of the foot wheel. Others recommend that it be placed sufficiently high so that there will always be one or two empty buckets directly below the one being filled. Others prefer to allow the ore to fall into the forward slope of the boot and be scooped up by the buckets as they come around. The last method saves considerable hight, which is important if the foot of the elevator must be placed in a pit, as is frequently the case. With a properly designed boot there is no danger of the buckets being torn off, although the ore fed may be as coarse as 2½ in.; if by chance a piece of ore wedges between the edge of a bucket and the boot, it is shoved upward until the increased clearance allows the bucket to pass, when the ore will slide down again, and will be taken up by the next or some following bucket. In many elevators there is no boot provided, the housing being simply extended down to the floor and allowed to fill naturally with ore, and sometimes there is not even any housing. An objection to scooping up the ore is the greater wear of the buckets, but this is offset by a greater wear on the belt when it is attempted to feed into the elevator so that the buckets will take the ore "on the fly." In order to avoid this wear of the belt, some mill-men insist that the ore shall be fed in from the side, so as not to strike the belt. The usual practice, however, is to feed directly toward the belt from the front. At the best, the ore spouted into the elevator is seldom caught entirely; there will be some that fails to be caught, which will accumulate in the boot and will still have to be scooped up.

An important feature in the design of a belt elevator is the arrangement of the discharge end, in order to insure a clean delivery of the ore. Some mill-men prescribe that the discharge chute should take off from the housing 10 in. below the center line of the head wheel; others prescribe 16 in.; others measure down 27 in. from the center of the head wheel shaft and then draw a line at an angle of 45 deg., the intersection of this line with the housing of the elevator being the point at which the discharge chute takes off. The last rule makes the elevator unnecessarily high. It is the practice of some mill-men to arrange the discharge chute so that a small quantity of ore will be retained

at the point where the stream strikes, in order to protect the bottom of the chute from abrasion; others arrange a special casting of iron for the ore to deliver upon, which is a very good device. There is always likely to be a small quantity of ore that will fall back into the elevator, and to avoid this dropping between the belt and the foot wheel, it has been found a useful expedient to insert a sloping partition at one or two places in the housing of the elevator, inside of the belt, so arranged as to discharge any droppings of ore into a small box outside of the housing.

In any form of belt elevator it is advisable to arrange the journals of both the head and the foot shaft so that the bearings will be as far away from exposure to dust and grit as possible. It is still better to arrange them so that a stuffing-box can be inserted between the bearing and the housing of the elevator. It is one of the bad features of the adjustable take-up boxes designed for the boot by most manufacturers that the bearings are arranged so closely to the boot that it is practically impossible to protect them from grit.

The power required to operate a belt elevator is comparatively insignificant. The apparatus itself is in equilibrium, and only such power has to be applied as is necessary to lift the ore and overcome the friction. The theoretical requirement of power to lift 10 tons per hour to a hight of 40 ft. is only 0.4 horse-power approximately, and allowing an equal amount for friction, the total is only 0.8 horse-power. If a horse-power costs $100 per annum (and in small plants in the West it is likely to come to about that figure), the cost of running an elevator for one year for power alone would be about $80. Assuming that this were the cost of operation for only 300 days of 20 hours each per annum, the quantity of ore elevated would be $300 \times 20 \times 10 =$ 60,000 tons, and the cost per ton would be 0.17c. Allowing 0.33c. per ton for repairs, which is a liberal estimate, the total cost of elevation would be only 0.5c. per ton. It is evident that there cannot be much objection to the extensive use of belt elevators on the score of cost. The chief objection is the loss of time that is likely to be suffered through their breakdown. This can be reduced to the minimum by the installation of a well-designed elevator of ample capacity and the best construction in the first place, and when in operation by a careful daily inspection and the making of necessary repairs at convenient times.

## TAILINGS ELEVATORS [1]

By W. H. Wood and E. J. Laschinger

(March 24, 1904)

On the Witwatersrand, owing to the flat nature of the ground, it is generally necessary to elevate the tailings from the mill, so that the pulp may flow into the vats of the cyanide annex. In cases where the mill is situated at a distance from the cyanide works, it is sometimes even necessary to elevate the mill product twice, in order to obtain the requisite grade for the launders. For this purpose tailings elevators of various kinds are employed.

The subject may be treated under a few distinct headings: (1) Tailings wheels, (2) tailings pumps, (3) bucket belt elevators, (4) air-lift pumps.

*Tailings Wheels.* — The tailings wheel is first in importance as being most generally in use on the Rand; there are quite a number of wheels of 60 ft. diameter, capable of raising the tailings from 200 stamps. Although the first cost of a large wheel is greater than that of any other elevator, its reliability, durability, and low maintenance cost have caused it to be generally adopted. Most engineers are so conservative that, when once a particular mechanism has proved itself satisfactory to perform a certain duty, they are loath to experiment with others.

As to the mechanical efficiency of tailings wheels, a test made on a 25-ft. wheel at the City & Suburban mine showed: Diameter of wheel, 25 ft.; lift of wheel, 19 ft. 1 in.; tailings and water lifted, 5549 lb. per minute; theoretical horse-power, 3.208; power actually used to drive wheel, with two intermediate counter-shafts, taken from motor, 8.847 horse-power; motor efficiency from actual trial, 75 per cent.; power delivered by motor, 6.935 horse-power; total power efficiency of wheel and driving mechanism, 48.51 per cent.

[1] Abstract from the *Journal* of the Mechanical Engineers' Association of the Witwatersrand, Jan., 1904.

It must be noted that as this is a small wheel, and as the driving mechanism involves three speed-reductions from the motor, the efficiency is probably much lower than it would be in a large wheel driven from a mill line-shaft with only a double speed-reduction.

The usual type of wheel on the Rand consists of an outer rim with a continuous inner flange on each side. The vanes are

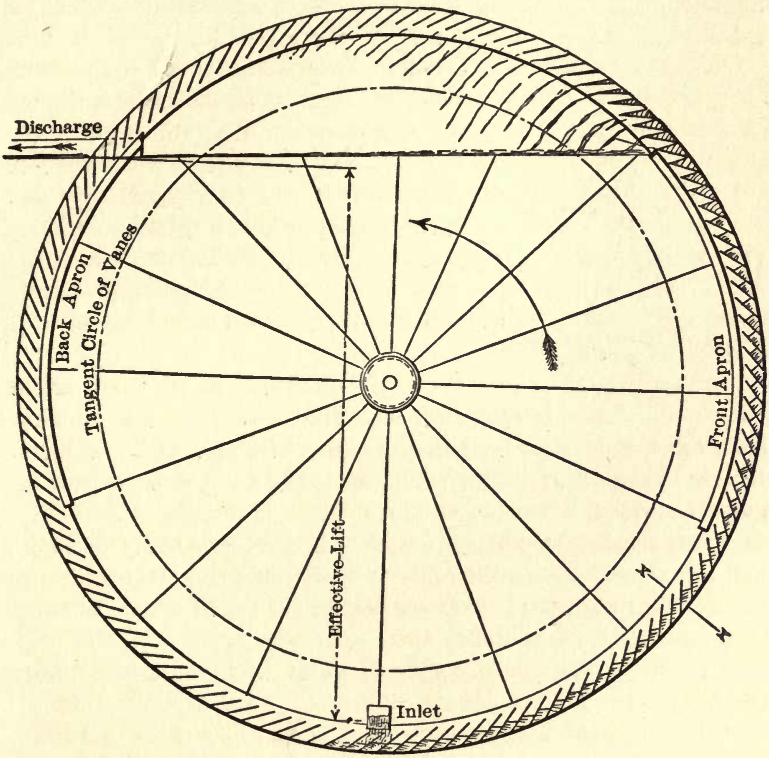



Fig. 35.—Sand Wheel.

straight and fixed to the inside of the rim, between the flanges. (See Fig. 35.) The outer rim is supported either by rigid arms or by tension spokes, arranged in pairs and splayed out to give lateral stiffness. The whole wheel is mounted on a heavy shaft and supported on bearings mounted either on masonry piers or on steel or wood structures.

The driving is accomplished either by belt, manila rope, or wire rope. It has been found, after many years' experience, that the method of driving the wheels at the Ferreira Deep, Henry Nourse, etc., has proved satisfactory, that is, by means of 1.75-in. diameter manila rope working around grooves on the rope run of the wheel, and driven from 11 ft. diameter cast-iron, ordinary type, grooved driving-wheel.

A point that is worth noting in rope-drives is that the grooves in the wheels must be at least 6 in. deep, a precaution which, if taken, will prevent the chance of the rope blowing off in high winds. As an actual proof of the durability of manila ropes on tailings wheels, we may mention that the rope at the Ferreira Deep was put on when the mill first started, over 4.5 years ago, and was replaced only two months ago, and that it actually worked for 3.5 years. This is a great improvement on any work done by the older type of belt drives, which, on the average, we believe, do not last more than about twelve months. Wire-rope drives have been tried, but so far without success. The new 60-ft. wheel at Knights Deep is driven this way, but it has not yet been working long enough for us to express an opinion.

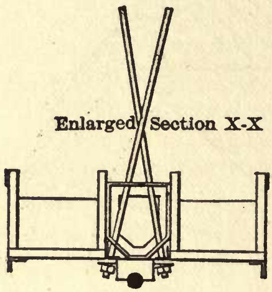



FIG. 36.

Until recently the construction was almost entirely of wood, but the latest wheels are built of steel, all except the lining of the rim and flanges, and the vanes, which are of wood, as being the material best adapted to resist the scouring action of the sand, and as being easily renewable in parts when broken or worn through.

The principle underlying the operation of these wheels is exactly the reverse of the old-fashioned overshot breast wheel. The liquid is introduced at the lowest point by a launder delivering into it, then raised up to a point near the top, where it falls out into a receiving launder as the vanes are inverted owing to the rotation of the wheel, thus spilling the liquid out. (See Fig. 35.)

The calculations necessary to determine the carrying capacity of a tailings wheel are rather complicated, and it is quite impossible to work out exactly on theoretical principles just what maximum

amount of pulp a wheel can handle. This is because it is impossible to say how much sand may remain permanently adherent to the vanes, and because the surface of the liquid in the buckets is constantly in motion. When designing tailings wheels for any certain size of mill, the following table is useful in arriving at the amount of pulp to be handled:

MILL PULP PER STAMP PER MINUTE

| RATIO, BY WEIGHT WATER TO ROCK | CU. FT. OF PULP PER MINUTE | LB. OF PULP PER MINUTE |
|---|---|---|
| 6 — 1 | 0.7796 | 53.472 |
| 7 — 1 | 0.9018 | 61.111 |
| 8 — 1 | 1.0241 | 68.750 |
| 9 — 1 | 1.1463 | 76.389 |
| 10 — 1 | 1.2685 | 84.028 |

One stamp is assumed to crush 5.5 tons per 24 hours.

A tailings wheel has its maximum carrying capacity when discharging at the level of its axle. That is, if pulp is to be lifted a hight of 20 ft., the wheel to lift a maximum quantity of material will have to be 40 ft. in diameter, but practical considerations such as those of cost and efficiency modify this deduction considerably. The cost increases at a greater rate than the ratio of diameter, and it is desirable, especially for high lifts, to discharge at a point as high up on the periphery as practicable, in order to keep down the size of the wheel. To meet this requirement the angle made by the vanes with the tangent to the periphery must be so chosen that the vane makes a small angle with the horizontal at the point of commencement of discharge. This is done by making a judicious selection in determining the diameter of the tangent circle of vanes.

The time to be allowed for the buckets to discharge their contents should not be less than 3 sec. for wheels, say, of 40 ft. diameter, and about 5 sec. for wheels of 60 ft. diameter; that is, the receiving platform should be long enough, or the speed of the wheel slow enough, that the buckets will take the stipulated time to travel over the receiving platform when they are in their inverted position in the upper portion of the wheel.

From theoretical considerations it is evident that, the larger a wheel, the greater the ratio of lift to diameter may be, and also that larger wheels may be run at a higher peripheral velocity

than small ones. It is evident also that a wheel could be revolved at such a rate that it would not discharge at all, but carry its contents completely round and round, owing to the centrifugal force becoming greater than the weight of the liquid.

The following table of "critical" or limiting speeds for wheels of various diameters will be of service:

| DIAM. OF WHEEL IN FT. | REV. PER MINUTE | PERIPHERAL VEL. FT. PER MINUTE |
|---|---|---|
| 10 | 24.218 | 761 |
| 20 | 17.125 | 1,076 |
| 30 | 13.983 | 1,318 |
| 40 | 12.110 | 1,522 |
| 50 | 10.830 | 1,701 |
| 60 | 9.887 | 1,863 |
| 70 | 9.154 | 2,013 |
| 80 | 8.562 | 2,152 |

The actual speed in practice is, of course, considerably less (generally about one-third of the critical speed, as given in the above table), but it is advisable to run at as high a rate as possible, in order to keep down the weight and cost of the wheel.

From a consideration of the strain diagrams for tailings wheels, it appears that the strain on the rim is independent of the number of spokes, and this is true within the limits of practice; it is also evident that the wheel with rigid arms is to be preferred to the wheel with tension spokes, because the maximum strains are only about half as great in the rigid armed wheel as in the spoke wheel. Practical considerations also point to the advisability of making the wheel with rigid arms, as it is easier to make it run true. The tension spoke wheel looks lighter and prettier, and may, perhaps, be slightly cheaper; but for durability and rigidity the heavy armed wheel is superior and more satisfactory.

The rim of a tailings wheel is generally the weak part in its design and construction, and, owing to the heavy and varying strain, it is more likely to give trouble than any other part of the wheel. It has been suggested by L. H. Lavenstein to make the vanes of a V shape, with the object of obtaining a cleaner discharge of the heavy sands which stick to the buckets. Mr. Lavenstein has also designed a spokeless wheel which consists simply of a stiff rim, with the usual buckets, guided by rollers, and driven by the friction of driving-wheels set under the rim of the tailings

wheel, which has continuous railway rails around its outer periphery. The object of this invention is to simplify the driving gear with its usual great speed reduction, and to save the cost of the heavy foundations or framework required to support the bearings of the ordinary wheel. The rim of such a wheel would, of course, have to be very strong and stiff to resist deformation if the wheel were of large size. We do not know of this type of wheel having been actually constructed and put into operation.

(2) *Tailings Pumps.* — Plunger pumps are used on some mines to elevate the mill pulp. They are usually made of the single-acting type, with long stroke and comparatively low plunger speed. The plungers must have clean water delivered under pressure behind the gland packing to wash off the sharp sand from the plunger on its return stroke and thus prevent the rapid grinding away of the plunger and gland. The greatest difficulty encountered is that of the scouring of the valve ports and valve chambers by the sands carried in the water. This is minimized, in pumps of good design, by making the ports as large as possible, thus keeping down the velocity of the pulp when passing through them. A certain minimum velocity of flow, however, must be kept up in order to prevent the settling of the pyrite and heavier sands. Removable liners in parts subject to scour are also provided.

A well-designed tailings pump is expensive, because of its size; it is required to handle so large a quantity of liquid that all the parts are necessarily massive. The chief drawback, however, is the maintenance cost, and, more particularly, the loss due to mill stoppage when repairs have to be made. The smallest item is generally the power cost; maintenance costs come next in importance, and the greatest loss is the reduction in gold recovery, due to stoppages of the mill.

While upon this subject it is well to point out how important it is that every plant should be provided with either a duplicate set of tailings elevators or some other alternative method of raising the pulp while the elevator usually in use is shut down for repairs. It might easily happen that the loss due to a single breakdown in a tailings elevator would more than pay for the installation of the most expensive kind of elevator known, as an alternative for doing this work in case of accident.

Centrifugal pumps have been tried, but those who have had

experience are best able to testify to the uselessness of the centrifugal pump for this class of work. The vanes may be scoured away by the sand in a few hours, leaving nothing but the hub remaining on the shaft.[1]

Plunger pumps and centrifugal pumps may be, and are, successfully used for elevating slimes, but when handling mill pulp the sharp grains of quartz cause so much scouring, necessitating so many and frequent repairs, as to render them but poor substitutes for tailings wheels.

(3) *Bucket Belt Elevators.* — Buckets fixed either to link chain or to rubber belts have been tried for elevating tailings. Owing to the sharp sand particles lodging in the moving parts or joints of a link chain, the links are soon cut through. In the case of buckets fixed to a rubber or composition belt the chief difficulty that seems to present itself is that of driving. The belt, being continually wet, slips on the pulleys, and if sufficient tension is employed to make the belt grip, the strain is too great and the belt tears apart.[1]

(4) *Air-lift Pumps.* — The air-lift or Pohle pump has recently come in for a great deal of discussion as to its adaptability, durability, and efficiency as a tailings elevator. That it can be successfully employed for this work has been demonstrated in actual practice. It is used in Australia in a number of mines for raising pulp to a small hight.

What its life may be is not so well known. The fluid must be kept moving at a rather rapid rate in this style of elevator, and the scouring action of the sands on the metallic pipes may be so considerable as to require frequent renewals of the piping. It would appear that thick cast-iron pipes would be best to use, or very cheap and durable piping might be made of wood, which has remarkable wearing qualities where scouring action is concerned. There are no valves or moving parts to give trouble, and at first glance this apparatus appears to present many commendable features.

The mechanical efficiency of this device is somewhat disappointing. Sufficient reliable tests have not, so far as we are aware, been made or published to enable one to predict with certainty what efficiency may be expected from any particular pump

[1] Both centrifugal pumps and bucket belts are extensively used for the elevation of tailings and other mill products in the United States. — EDITOR.

operating on this principle. Some tests on the Pohle pump for lifting water have been published.[1] From these records it appears that the efficiency is not high (about one-half that of a pump), and varies with a good many factors, such as ratio of submersion to lift, size of air pipe, hight of lift, etc.

The whole question of tailings-elevator arrangements may be summarized thus: Install the most efficient, durable, and reliable elevator to do the regular work, even if the first cost be high. Install a stand-by which is, first of all, reliable, but as simple and cheap as possible; whether it is efficient or inefficient is practically immaterial.

We wish, however, to point out that if a really well-designed tailings wheel is installed, it is doubtful whether there is any necessity to have a stand-by apparatus, as the satisfactory record of the tailings wheel at the Ferreira Deep illustrates. But if in a large plant two or more tailings wheels are required, and they are not far distant from each other, it will be advisable to install a cheap stand-by, as the risk of breakdown increases in direct proportion to the number of elevators required. The stand-by should then be so arranged as to do the work of either or any one of the tailings wheels.

[1] "Mine Drainage, Pumps, etc.," H. C. Behr, p. 170, *Engineering*, Aug. 17, 1900; and paper by William Maxwell, British Association of Waterworks Engineers, *The Engineer*, Aug. 14, 1903.

## TAILINGS ELEVATORS

BY R. GILMAN BROWN

(April 14, 1904)

Apropos of the interesting paper by Messrs. W. H. Wood and E. J. Laschinger, published in abstract in the *Engineering and Mining Journal* of March 24, the following notes concerning different devices for lifting stamp-mill tailings may be of general interest. They are the result of experiences in the 20-stamp mill of the Standard Consolidated Mining Company, at Bodie, California:

*Air-lift.* — The problem was to raise 60 tons of the quartz sand and about 90,000 gallons of water per day 45 ft. This was equivalent to about 9.4 cu. ft. per minute, weighing 604 lb. The theoretical horse-power in consequence was less than 0.85. The efficiency of an air-lift is dependent upon the ratio of submersion to lift, and accordingly a double lift was determined upon, to avoid expense in obtaining submersion. A shaft was sunk under the mill and a 10-in. pipe placed in it, closed at the bottom. Within this a 3-in. lift-pipe was placed. This pipe gave a lift of 22.5 ft. above the top of the 10-in. pipe, and discharged into another 10-in. submersion pipe, with 3-in. lift as before, giving 22.5 ft. of additional lift. The air supply was from a Garden City rotary blower, capable of displacing 188 cu. ft. of free air per minute and delivering it at 10 lb. pressure. The air-pipe was 1 in. and came down outside the lift-pipe, turning up a few inches within the lower end. This device operated with fair success, but when part of the mill was idle, the lowered velocity in the lift-pipe allowed the heavier particles to settle against the rising current and in time blocked the pipe. No tests for power consumption were made, but the efficiency would be at best poor. I have recently been using an air-lift for unwatering a mine under conditions where pumping was impossible, and have found everything satisfactory in its use, except the efficiency.

This was about 12 per cent. measured between the electric meters, measuring the current which operated the air-compressor and the actual water thrown. The air pressure was, however, 85 lb.; in the case of the Garden City blower, giving but 10 lb. pressure, the efficiency would be better. It should be added that this efficiency was for the case of the submersion being equal to lift.

*Bucket Elevator.* — This was also tried, but so much sand adhered to the buckets that, apart from the wear on the links, the system was not satisfactory, particularly during cold weather.

*Centrifugal Pump.* — This was used for a year or so for this service, notwithstanding the heavy wear. The consumption of metal from this cause was lessened by various expedients, but probably at best it amounted to 4c. or 5c. per ton of sand handled. Added to this was the high power consumption. The mill was operated by electricity, and meter measurements made at the main motor showed 15.5 horse-power as the consumption of the centrifugal pump, including belting, line-shaft running, etc. This may be considered by far the most serious drawback to the centrifugal pump for this work. I believe that the consumption of metal can be still further lessened by changes in design.

*Frenier Pump.* — This somewhat anomalous machine has been doing excellent service and has shown no excessive wear, the chief wearing part being the stuffing-box in the axis. The power consumption is less than one-third that of the centrifugal pump. The speed must be closely adjusted to get good results, and the pump is sensitive to sudden increase or decrease of flow delivered to it, change in either direction causing the supply-box to overflow until the pump has adjusted itself to the changed condition. For ordinary work in small units it is by long odds the best of the machines enumerated above. As 45 ft. is an excessive lift for these pumps, we used two 54-in. pumps, one lifting to the other.

# PART VI

# DISPOSAL OF TAILINGS

# A SYSTEM OF HANDLING SAND MECHANICALLY FOR CYANIDE VATS[1]

By Charles Butters and Albert F. Crank

(December 5, 1903)

The Virginia City works of Chas. Butters & Co., Ltd., are situated on the eastern side of a cañon leading down from Virginia City, and at a point about three miles distant from that town. This site was chosen, close to a large heap of tailings, in order to conveniently treat not only these tailings, but also ore from the Comstock lode and other veins in the vicinity. An inclined tramway with two lines, worked by a wire rope, is carried westward from the works across the cañon by a wooden trestle bridge, and continued for 1200 ft. along the length of the heap. The tailings are loaded by horse scrapers direct into the tram-cars, for transport to the works.

As there is a fall of 600 ft. between the mean level of Virginia City and the works, tailings can be sluiced down by a small stream of water from any heap in the camp, along a 10x10-in. wooden flume, direct to the classifier in the works. About three miles of main flume and a mile of branch flume have been built, and are now in successful operation.

The tramway and flume enable tailings to be conveyed to the works at an exceedingly low cost per ton, as the amount of shoveling is reduced to a minimum. The next point is to be able to handle the sand economically between the collecting and leaching vats, and discharge it to the waste heap.

The works are designed principally for the treatment of slimes, which make up 55 per cent. of the total amount of material treated, the remaining 45 per cent. being sand. Consequently, as 275 tons are treated daily, 124 tons of sand have to be dealt with. Labor in this district is scarce, and workmen command

[1] Abstract of a paper read before the Institution of Mining and Metallurgy, London, Nov. 19, 1903.

high wages; therefore, supposing hand work to be adopted, the capacity of the sand department would be governed by the amount of labor available for shoveling, while the value of the tailings that could be worked at a profit would in part depend upon the expense of this labor.

The machines adopted to do the work are portable excavating and distributing appliances, capable of dealing with damp or drained sand. They are the invention of Hiram W. Blaisdell, of Los Angeles, Cal. It was claimed that they would discharge, transfer, and distribute more sand in a given time than was possible by hand labor, even when mechanical conveyors were employed. It was further said that all the labor in the sand-house could be done, according to the size of the works, by one or two attendants. Lastly, it was claimed that, owing to the thorough mixing during the process of excavation, and to the subsequent light and even distributing of the material in the vats, the cyanide solution penetrated more rapidly and leached more completely than in any light system now in vogue. As the Blaisdell machines are new factors in cyanide practice, it is well to describe them fully, and to state the results obtained, as the matter is sure to be of interest to mining engineers engaged in similar operations.

From August 4 to August 7 of this year a series of records were taken of daily performance under regular working conditions. The results obtained, together with the records of the past six months, confirm the original estimate of the value of the system, and substantiate the claims advanced by the manufacturers. At the present time, the amount of material treated daily in this department is 250 tons — that is to say, 125 tons are handled twice in that period. Under the direction of the manager of the works, the shiftsman on the sand tanks makes the required changes in pump connections, valves, and solutions, and operates the excavating plant in a working day of 10 hours.

The accompanying plan exhibits the general arrangement of this sand-handling plant. It consists of a Butters distributor, for charging the collecting vats; one Blaisdell bottom-discharge excavator, for discharging all the vats; one Blaisdell centrifugal distributor, for charging the vats; and a combination of four 16-in. belt conveyors, for transport of the sand.

There are eight wooden tanks, or vats, each 30 ft. diameter

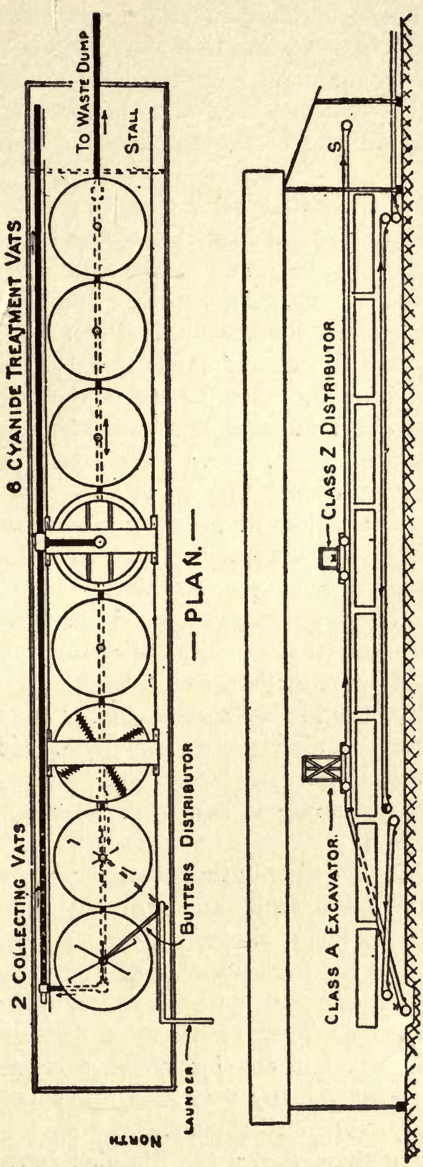



FIG. 37. — Diagrams of Virginia City Plant for Treating Sands.

by 6 ft. deep, intended for 125 tons of sand each; they are set upon one continuous foundation. Each vat has a central discharge gate, with chute leading to a conveying belt below. The two north vats are used for collecting and draining the sands received from the tailings wheel, and their contents are subsequently removed to one of the six leaching vats for cyanide treatment.

Conveyor No. 1, located below the two collecting vats, is approximately 65 ft. long, and conveys to the left, or northward. Conveyor No. 2 is 20 ft. long, and conveys the discharge of No. 1 at right angles, delivering to conveyor No. 3, also at right angles. At the north end, where the discharge of No. 2 is received, conveyor No. 3 has an inclination of about 11 deg. for 67 ft., beyond which it runs level for 123 ft. to the end of the line of vats. By this incline the sand is brought from a level 4 ft. below the bottom of the vats to a level 18 in. above their tops. In ground plan this conveyor is parallel with the line of vats, with a distance of 17 ft. 8 in. between center lines. A tripper extends from the Blaisdell distributor and diverts the load from conveyor No. 3 to a short conveyor upon the machine. Conveyor No. 4 is placed below the discharge gates of the six leaching vats, and is arranged so that it can be run in either direction. When running north (or to the left), the discharge from leaching vats is delivered upon conveyor No. 1, and then carried around the system to the distributor. When running south, the discharged tailings are carried to the waste heap. In place of No. 4 conveyor, a launder leading to the waste pump is employed when single-treatment sand is being worked.

The excavating and distributing machines are movable steel structures, spanning the tanks and traveling upon 16-lb. rails placed 9 in. outside, 12 in. above, and extending the length of the line of vats. It has been found, with the use of this system, that the pulp is laid down so perfectly in the vats that no benefits were derived by the use of a second treatment; and, although the plant was laid out for giving a second treatment, or several treatments, to any vat, these have been found unnecessary and have been discontinued.

As employed in these works, the Blaisdell excavator consists of a steel truss bridge, supporting at midspan the excavator and driving gears, and at one end the motor and traverse gears.

The bridge deck is 33 ft. over all in length and 5 ft. 6 in. wide. The hight of the truss is 10 ft. The bridge is supported upon trucks, each of which contains two 15-in. flanged wheels, 10 ft. between centers. The span or distance between rail centers is 31 ft. 6 in. The midspan vertical truss bridge carries cross-head guides, together with the heavy screws for raising and lowering the cross-head. The latter contains the upper bearing of the spindle, and a thrust-box supporting the weight of the spindle, excavator beam and disks. A heavy frame in the bridge floor carries a bearing, in which rests the main bevel gear-wheel, driving the excavator spindle. This gear serves also as the lower guide for the spindle, and is centered below the cross-head bearing. Power is transmitted to the spindle by steel keys secured in the gear, and entering keyways cut the full operating length of the spindle. Securely keyed to the lower end of the latter is a cruciform beam of structural steel parts.

The total over-all length of the two long arms is 29 ft. 8 in., thus allowing a 2-in. clearance when operating a 30-ft. vat. The short arms have an over-all length of 11 ft. 4 in. Bolted to the under side of these arms are cast-iron hangers of graduated lengths, each terminating in a bearing, wherein revolve hardened steel spindles projecting from the center of the steel excavator disks. These bearings are so designed that sand may not enter.

The disks are placed obliquely to the radial lines. The angle varies from 15 to 25 deg., the large angle bearing nearest the center. Twelve disks about the center are flat plates, while all other disks are dished with spindles projecting from their side. The concave side is presented to the work. There are in all 46 disks, so spaced from the center that the position of each disk upon its respective arms falls upon a spiral generated from the center, in the direction of rotation of the beam. A radial section of the contents of a vat during excavation would show a series of 36 surface undulations. The hollow of these waves will rest upon a line having a fall toward the center of about 1 in. in 12 ft.

The angular setting upon the arms presents the edge of the disks to the work as a cutting line. As this advances, the sand is cut with a rolling and nearly vertical motion, with little friction. The line of cut extends in a diagonal across the furrow, and a shearing rather than scraping action is secured. The dished

shape of the outer disks gives the necessary lead and clearance to the cutting edge. As the disks revolve only when in contact with the sand, it follows that there is no friction due to drag, so that the power over and above that required to overcome internal machine frictions and the insertion of the cutting edges is applied to the actual moving of sand. This is raised by the revolving disks until dislodged by gravity, when it rolls from the dished surface into the path of the following disk. The weight of the beam, with spindle and disks, is sufficient to force the disks into the sand, but the depth of cut is regulated by the feed-screws. These receive automatic motion, or feed from the bevel gear pinion-shaft by means of a pawl, ratchet-wheel, and a gear-chain. The pawl makes 25 strokes per minute, and may be set as desired to feed from one to seven teeth upon the ratchet-wheel with a beam revolution of five per minute. This produces a vertical motion of the excavator spindle of from 0.02 to 0.135 in. per revolution. The range of adjustment allows the work of excavation to be pushed to the maximum limit permitted by the firmness of the sand or the capacity of the conveyors. There is, in addition, a belted connection between the ratchet-wheel shaft and the main shaft, by which a vertical speed of 14 in. per minute may be secured for rapid raising after excavating or for adjustment before a run.

An automatic device upon the cross-head guide may be set to stop the excavator at any desired depth, and a similar device in a permanent position at the top of the guides prevents injury from raising the beam too close to the bridge.

Power for the machine is taken from two overhead wires by a trolley, and supplied to a direct-connected motor upon the bridge, where is also located the switchboard panel, containing the rheostat with reversing switches and fuse-block. The motor is back-geared to a line of shafting extending to the center of the bridge. From this shaft, power is taken to operate the traverse gear train for the following four purposes: Shifting from vat to vat, operating the boring bar, rapidly raising and lowering the feed, and driving the counter-shaft of the excavator spindle gears.

In dealing with a filled vat, the boring head is first brought over the opened central discharge valve and driven through the sand. The excavator beam is then raised to clearance position, and is brought over the discharge opening by moving the bridge

forward. The beam is now lowered until the disks meet the sand, when the revolving gears may be started and the required feed set. The machine then runs without attention until it comes into contact with the automatic stop.

The Blaisdell centrifugal distributor is supported at midspan of a movable steel bridge having a deck and trucks similar in design to those of the excavator. It is not as heavy in construction, and the trussing is secured by suspension rods and spreader below the deck. At one end of this bridge a tripper is placed, overhanging conveyor No. 3, and diverting the sand from the latter to a conveyor upon the bridge deck. This discharges into a cast-iron hopper at the bridge center. A revolving vertical spindle passes through the hopper and has a horizontal steel disk keyed upon the lower end. Riveted to the upper surface of this disk are short radial vanes of angle iron. This distributor and spindle are supported by bearings above the hopper. Pedestals extend upward from the hopper, and contain the bearings mentioned and those of the bevel-gear shafts. The pinion-shaft of the latter is belted to the head shaft of the bridge conveyor.

The vaned disk, or distributor, revolves rapidly, and sand falling from the hopper upon the whirling surface is showered evenly over the vat surface below. Power for the distributor is taken from the overhead wires in similar manner as for the excavator. The speed of revolution may be changed by both cone pulleys or rheostat.

A cylindrical guard-ring of light sheet-iron, 15 in. wide and 24 ft. in diameter, surrounds the distributor. It is supported upon brackets extending from the bridge, and has a 3-in. clearance over the side of the vats. This ring prevents the loss of stray sand particles when filling near the top of the vat. When working the distributor, the shiftsman starts the traverse gears, and runs the distributor into position over the closed central valve of an empty leaching vat. He then starts the distributor and conveyor. Leaving the machine, he passes along the sand-house to the switchboard of the motor driving the conveyors. After starting the conveyors, he passes on to the excavator, which he has previously placed in position. He now starts it, and after watching for a few minutes, to see that the feed of excavation is properly set, he leaves it to itself.

This machinery, during the present year, has demonstrated

that the work in the sand-house may be executed regularly in a given time entirely independent of labor conditions. Both the supply and cost of labor may be disregarded, for the primary consideration has now become merely the cost of power. It has been demonstrated that the cost per ton for handling a given quantity of sand between the tailing wheel and waste dump, once or oftener, may be determined in advance to a nicety. The tests show that the cyanide manager has in the excavator a valuable instrument, which is of service also in understanding correctly the conditions existing in the leaching and collecting vats. The power required to discharge a vat may be quickly determined by the readings from the volt and ammeter. Comparison between this reading and a standard table of reading will show the degree of firmness with which the sand is packed in the vat.

An uncalled-for rise in the power used under regular working conditions will call attention to a possible increase of the percentage of slimes or fine sand present, or to incomplete draining. The compact nature of sand in a collecting vat is well known. In these works, as already stated, treatment is not carried out until the sand has been removed from the collecting vat and transferred to the leaching vat. One advantage gained is that the subsequent percolation is exceedingly sure, rapid, and thorough; 125 tons of sand will, when drained, form a deposit of 3 ft. 6 in. deep in a 30-ft. collecting vat. After the complete disintegration secured by the excavating process, the sand is again broken up and blended by the action of the distributor, with the result that the 125 tons will produce a depth of 5 ft. 8 in. in the 30-ft. leaching vats before the introduction of solution, and subsequently settle 8 in. during treatment. Or 1 in. depth of sand in the collecting vat will produce 1.62 in. in the treatment vat before percolation, and 1.42 in. afterward. The opportunity secured for complete percolation is obvious.

Those familiar with the chlorination process will better understand the condition of this sand when I state that the loose condition which is secured by screening the charge into a chlorination tank is here obtained by means of the Blaisdell distributor. A chlorination tank, 20 ft. in diameter, will be filled to a depth of 7 ft. 3 in. by screening in an 80-ton charge of roasted concentrates. This depth decreases 8 in. by subsidence during treatment.

The charge will therefore weigh 70 lb. per cu. ft. before and 77 lb. after treatment. With the distributor at Virginia City, these weights are 63 lb. and 71 lb. respectively. In the first instance, the contents settled 10.1 per cent. of the original depth, and in the second instance 11.7 per cent. It appears from this that there is less packing from the distributor action than from the screening method, which has been hitherto the most perfect known.

At Virginia City the yearly cost is $84 per electrical horse-power, or 0.96c. per horse-power hour. An addition of 15 per cent. should be made to cover the loss in the motor-generator set. This brings the cost up to 1.1c. per horse-power hour.

The total power used by this system for the double handling of 125 tons a day throughout the sand-house creates a daily expense made up of the following items:

| | | | | Cents |
|---|---|---|---|---|
| Discharging collecting vat | 30.5 | h. p. at | 1.1c. | 33.55 |
| Conveyors, Nos. 1, 2 and 3, | 41.5 | " " | 0.96c | 39.80 |
| Distributing | 8.5 | " " | 1.1c. | 9.35 |
| Discharging leaching vat | 11.5 | " " | 1.1c. | 12.65 |
| Total cost for 125 tons | | | | 95.35 |
| Total cost for 1 ton | | | | 0.763 |

It has been estimated by the Blaisdell company, the manufacturers of these machines, that wear of disks and parts will amount to $50 per annum, and an estimate based upon the past six months' use shows that this amount is correct. To this may be added $30 for grease, waste, oil, etc., required by the entire plant. This, then, would give a total of $80 per annum for maintenance, or 21.9c. per day.

The daily working costs may be tabulated as follows:

| | Cents |
|---|---|
| Power | 95.35 |
| Supplies and wear | 21.90 |
| Labor | 60.00 |
| Total | 177.25 |
| Cost per ton | 1.415 |

The total daily cost for all handling, changing of solutions, and general sand-house work is the cost of power and maintenance, as shown, plus the total time of the operator ($3 per day), making $4.17, or 3.336c. per ton.

If the works had to be enlarged from a capacity of 400 tons daily to a capacity of 1000 tons, and the percentage of sand continued about what it is now, the only change required for enabling the machines to handle the 480 to 500 tons of sand twice in the 24 hours would be to replace the present motor upon the excavator by one of greater horse-power, allowing a greater feed upon the collecting vats. There is no reason why an excavator should not be made sufficiently strong to discharge 100 tons an hour from a collecting vat. It would, of course, have to be made heavier than the present machine, which is intended for the present duty, and which has fulfilled all requirements.

The introduction of the Blaisdell excavator marks a new era in the construction of cyanide plants, and is so important that it will be worth while to introduce it even in the case of existing sand plants which admit of the necessary modifications. The reasons for its use are strong — the saving of gold by higher extraction, and the reduced cost of labor.

El Oro Mining & Railway Company, of Mexico, is now erecting a complete cyanide plant for the new 100-stamp mill. The portion of the works for treating the sands consists of nine collecting vats, 22 ft. by 10 ft., served by one Blaisdell excavator, and of 14 leaching vats, 40 ft. by 6 ft., in two rows, served by one excavator. By arranging a series of Robins belt conveyors, the contents of any one leaching vat can be transferred to any other leaching vat, or conveyed direct to the tailings dump; and the capacity of excavators and of the belt conveyors is 100 tons per hour. The estimated cost of working this plant in Mexico, at the rate of 200 tons per day, is 3c. per ton, and it requires the services of two men, the cost of power being taken at $10 (American) per horse-power per annum.

The conditions under which the tanks are filled are so perfect that only one treatment is required for complete extraction, as is the case in a chlorination tank, because the sand is in the same condition that it would be if it had been screened into the tank.

No cyanide solution is ever run into the collecting tank, as its contents, after having been filled from the battery, are allowed to drain, and are then transferred by means of the excavator directly to the leaching tank. Therefore, no loss of gold can take place from cyanide solution having been introduced into the collecting tank, as is the case when the collecting tank is filled

directly from the battery, and gets its preliminary treatment of cyanide solution in the same tank.

At Virginia City, as much as 50 per cent. of gold was found dissolved out of the charge, by filling into a collecting tank in which the previous charge had received a preliminary cyanide treatment, and, strange to say, this loss continued for over two weeks after the use of cyanide in the filling of the vat had been discontinued. While no trace of cyanide could be obtained at any time, still the loss went on, until after two very anxious weeks it gradually stopped. This occurred on two different occasions. At first it was attributed to an excessive amount of ferric salts in solution, acting possibly as a solvent for gold; but it was found later that it was from the traces of cyanide that could not possibly be detected by any chemical test.

Mr. Hennen Jennings and Mr. Butters had both made a series of determinations at Johannesburg as to the gold in the battery water from this source, and with similar results — gold was invariably found. It is not easy to state the exact loss from this source, but that it is a source of loss which should be eliminated, I think all engineers and metallurgists will agree. The use of the Blaisdell excavator reduces the number of vats required to produce a given result, even with the separate collecting vats, as rapid percolation takes place in all the vats used for leaching, while at present only one-half the vats are in a condition for perfect percolation.

The elimination of all outside labor from the works will naturally appeal to every one, as the shiftsmen with this apparatus can transfer the contents of sand tanks and discharge them when they are in order, and need not wait until other workmen are ready. Much time is saved, as the plant is much more elastic, and nothing waits or depends upon hand labor.

## ECONOMY IN MILL WATER

By Jesse C. Scobey

(December 10, 1903)

With no running or surface water in sight or available, nine Wilfley tables were operated, during the year 1902, at Washington, Arizona, in a plant treating 100 tons per day, and requiring a 150-horse-power steam equipment.

A water supply was accumulated in a large storage tank before starting the plant, and this water, when the plant was running, was in constant circulation, being alternately fouled and cleaned of both slimes and acid. To prevent any consumption in excess of the normal allowance was vital, as it would eventually close the mill. The reserve supply generally sustained the minor internal losses, and only on a few occasions was it necessary to shut down from an actual lack of water.

The final improvements, adopted from experience, gave a saving of 97 per cent. of the water in circulation, and, in view of the 20 per cent. loss admitted to be made in the South African cyanide plants, this loss of only 3 per cent. may well be reviewed. The comparison in no way criticizes the South African plants, as the widely different treatments account in a large way for the discrepancy. Mr. Denny said recently, in the *Engineering and Mining Journal*, that the discharged tailings retain only 10 per cent. water, and in this he means the sands only, else, of course, he would not have the 20 per cent. loss; this construction puts the entire penalty upon the separation of the slimes, as in my saving of 97 per cent. the sands carried 25 per cent. water. The stamp-mill practice produced more slimes than did our roller mill, both from the manner of crushing and the method of screening, using in the stamps 30-mesh, and on the trommels for the rolls a 12-mesh cloth.

That the 20 per cent. loss is excessive is admitted, when it is proposed to reduce this to 10 per cent. by a better system for

saving the slimed water, in the use of 20-ft. circular vats, which step is, to my mind, a proper one. So far, the system proposed is weak in the vital point of discharge, which must be over the entire periphery and must not be drawn from one point.

The entire water supply at Washington was obtained by sinking a well, from which was run a drift across and under an arroyo, draining an extensive water-shed. Tests showed that this well would supply 7500 gallons per day in the dry season; of course, more in the wet season, but that did not affect the problem. This amount was considered ample when properly used, and a small boiler and pump were installed with a 2-in. pipe line, to deliver this water to the plant, 1½ miles distant.

At the time I took charge of the plant, all machines were in place, and the design of the building was such that they must be connected up as they stood. In brief outline, the ore carried copper, lead, and zinc in a silicious gangue, heavy with garnet. It was crushed in rolls to 12-mesh and roasted; after cooling, it was treated magnetically, and about 20 per cent. of the ore was lifted as a copper concentrate, which was reduced in a reverberatory furnace to copper matte. The remaining lead, zinc, and garnet tails were submerged, and treated on tables to a lead-zinc concentrate, a zinc-garnet middling and tailings. Each of the two metallic products was redressed on two tables to shipping lead, and to zinc suitable for drying and subsequent magnetic cleaning. The nine tables for this work required, in feed and wash water, 900 tons of water per day, or 216,000 gallons.

The table floor was well laid in hydraulic cement, draining to the center, through which ran an open cement launder, that carried the tailings from the first five tables, and the waste water from the four dressing-tables. About 5 per cent. of the ore was taken out as a lead concentrate, which was collected in special boxes, from which the overflow water was discharged, after passing under and over baffle-boards, on the cement floor, from which it drained into the main artery. The concentrates were shoveled from the boxes into hemp sacks, and when first sacked would contain 25 per cent. moisture; these drained to 15 per cent. moisture in standing, before being finally removed to the shipping platform. The 15 per cent. moisture in 5 tons of lead concentrates made a loss of 0.75 ton or 180 gallons per 24-hour day.

The zinc dressing-tables gave about 15 per cent. of the ore as material to be dried for final magnetic work. This was collected from the tables to an automatic drag, similar to the one about to be described, in which the sands were separated, giving waste water to the tailings launder, and the zinc, with 25 per cent. water, to the drier, where the water was lost in evaporation. The 25 per cent. moisture in 15 tons of zinc concentrates made a loss of water of 3.75 tons, or 900 gallons, per day. This, with the loss in the lead, left 214,900 gallons discharging through the main launder, with recovery of which we were mainly concerned.

The extraction of copper, lead, and zinc left about 60 per cent. of the crude ore to be carried in suspension, by this water, to a cement pit, from which it was raised 20 ft., a head sufficient to allow the sands to be drawn off and the cleared water to be returned by gravity to the head-tank which supplied the tables.

Fig. 38 shows the means of accomplishing the separation of the sands from the slime water. The sand-box is in the form of a truncated pyramid inverted, with the larger triangle at the top, having a 7-ft. base and 10-ft. altitude. A 5-ft. depth of water filled the box to within 4 in. of the top. All sides of the box slope at an angle of 60 deg. toward the bottom of the box, or toward the edge up which the sands are dragged at an angle of 30 deg. In this way all material settled in the box is deposited under the drag belt. When full, the settling area of the surface of the water is 35 sq. ft. The tailings from the elevator are discharged into the sand-box at *A*, where the false back *a* extends across the box and to within 18 in. of the bottom, allowing a 6-in. space for the full width of the back, which conducts the sand and water to the bottom of the pyramid, where it is practically entered at the apex, and under the bottom pulley, around which the drags are carried. The rising current is thus subjected to an enlarging area, increasing as the square of the distance ascended, which accordingly reduces the velocity in like proportion, and effectually drops the sand.

From the surface the overflow water and slimes are taken from the box by the 6-ft. weir B over the edge away from or opposite to the sand discharge. This wide discharge materially reduces any tendency to form currents. The drag-belt *C* is 12-in. five-ply rubber, traveling 15 ft. per minute over the foot pulley *z* and the head pulley *c*. The head shaft is in fixed bearings,

to accommodate the gearing necessary to drive at the slow speed, while the foot shaft is held loosely in its position by wooden bearings, as is the practice in the Joplin mills. These bearings are in the water at the bottom of the box and are in no way

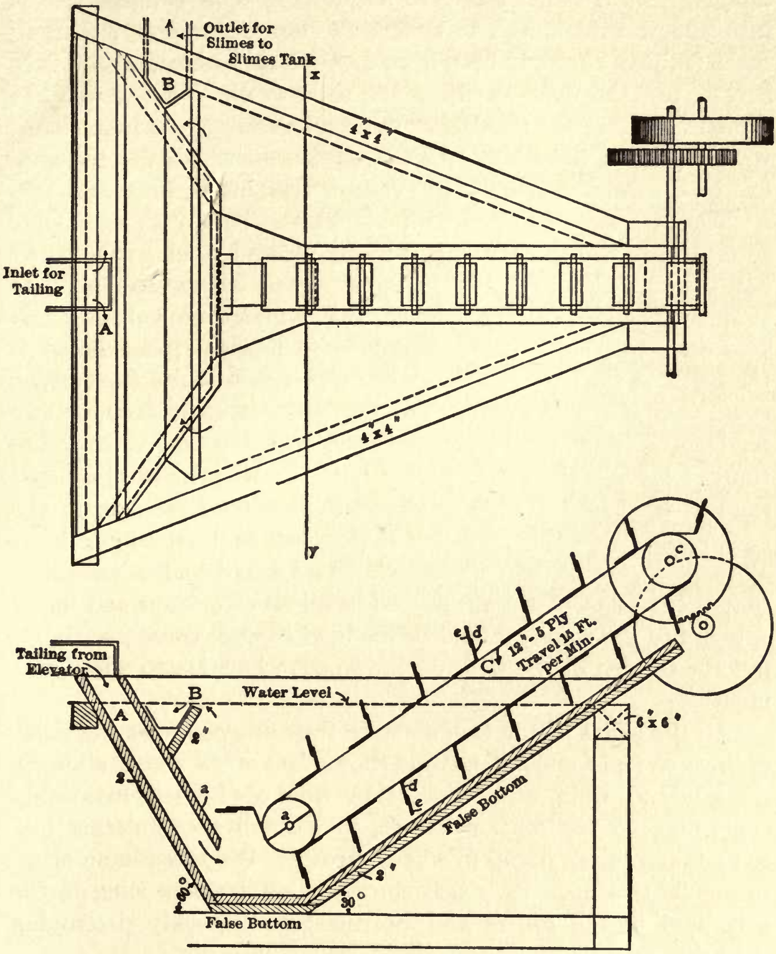



Fig. 38. — Sand-box in Water-saving Plant, Washington Mine.

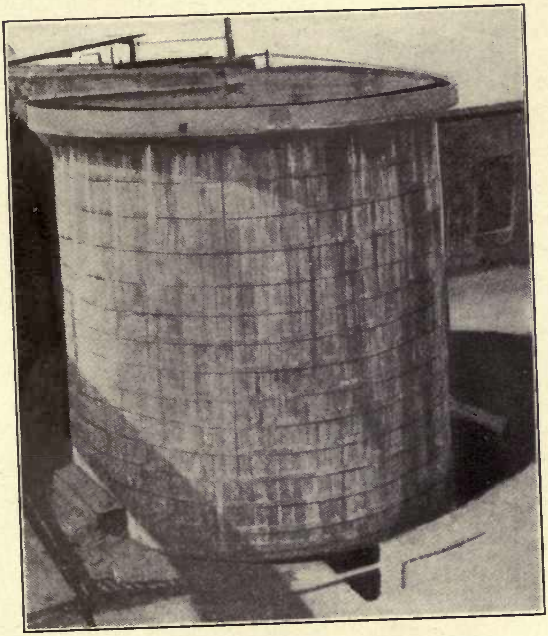
protected; they consist simply of a half-round cut in the end of a 4x4-in. hard-wood block. These pieces are held in place, each, by two bolts in slotted holes in the hickory, with just sufficient tension to keep the belt taut, and at the same time enabling

it to be taken up by slightly tapping the block down with a sledge. Our experience with this kind of bearing in such a place, or in the foot of wet elevators, has been to prove its superiority over all iron and babbitt bearings, arranged either bridged or overhanging. It is much easier to admit that sand is going to get into your bearings, and to design the box accordingly, than to make fruitless efforts to keep it out. Under the above, apparently severe, test the wear on either the shaft or the bearing was not disastrous, after six months' continuous running. A hole closed with a plug in the bottom of the box, from which the contents can be drawn, is convenient in case of clogging or breakage.

The belt was equipped with brackets *d*, bolted 14 in., center to center, to which the drags proper (*e*) were bolted, with slotted holes in the plate, for taking up, reversal, or renewal, without changing the more costly bracket, upon which there is little wear. The drag was constructed especially to leave a space between the belt and the plate, so that when leaving the water the current could escape over, as well as around the sides, thus lessening any tendency to wash away the sand carried on the drag. From the time of leaving the lower pulley until discharging the gathered sand at the top, the edge of the plate rested on the floor of the edge of the pyramid, up which it traveled, protected by a 2- by 12-in. piece, that could be renewed. Acid water made it necessary that the bracket and drag, as well as all elevator cups and metal exposed to the water, should be made of copper; consequently we put the burden of the wear on the wood, which lasted about two months.

All the sands of the tailings were thus dragged over the point of the box, and some 2 ft. beyond the surface of the water, allowing the sands to drain, after which they were discharged, invariably containing 25 per cent. moisture, to a small accumulating bin, to be taken to the dump in wheelbarrows. We failed in an effort to handle this mass by a belt conveyor, as it would cling to the belt, fouling the rollers and bearings, and quickly destroying the belt.

The water delivered from the sand-box carried about 10 per cent. of the crude ore treated, in slimes, while the sand delivered was 50 per cent. of the ore. The loss of 25 per cent. moisture, in the sands going to the dump, amounted to 12.5 tons of water or 3000 gallons per day. This loss now reduced the quantity



TANK AT WASHINGTON MILLS, ARIZONA.

going to the slimes tank to 211,900 gallons, contaminated by 10 tons of fine slimes.

A photograph shows the tank in question, which was simply a 16- by 16-ft. standard round tank, to which has been added a false lining that gives the interior the appearance of an inverted cone, with sides 60 deg. from the horizontal, truncated at the bottom of the tank in a circle of 2 ft. diameter. This lining is essential, as otherwise the settled slimes would not feed regularly to the discharge. No especial care is needed in putting in this lining, as it is supposed to fill behind with water, and eventually with slimes.

Around the upper rim of the tank, about 10 in. down, is fitted a 10-in. circular launder, enclosing the whole circumference. This was most expeditiously made by forming the bottom of the

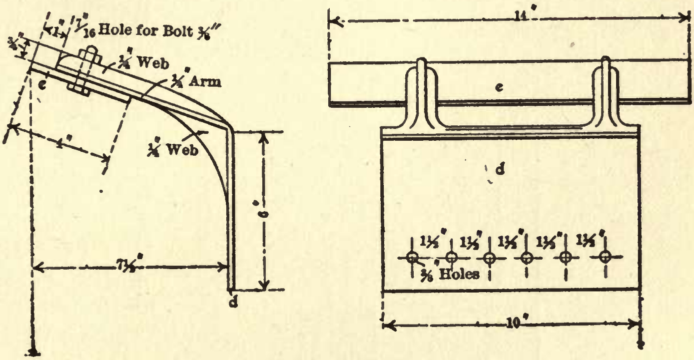



FIG. 39. — Details of Scraper.

launder from sections sawed from a 2-in. plank to fit the outside of the tank. After securing these in place temporarily, the outside of the launder was formed by tacking to the 2-in. bottom the lower edge of a 10-in. six-ply rubber belt, which was easily formed around the tank in one piece. This was quite stiff enough to maintain itself erect, especially when the whole was incased by an ordinary hoop-band with draw-nuts, which was so placed as to draw the belt against the 2-in. bottom, and the whole securely against the tank.

It is essential to have the overflow water drawn evenly over the top of the tank from all parts of the circumference, and, as the tops of the staves are irregular, and hard to dress to a plane,

this is facilitated by nailing around the top a 0.5-in. batten on edge, so that about one-half of its width stands above the ends of the staves; this thin strip is easily bent around the tank, and its upper edge can be then planed down to an exact level, when

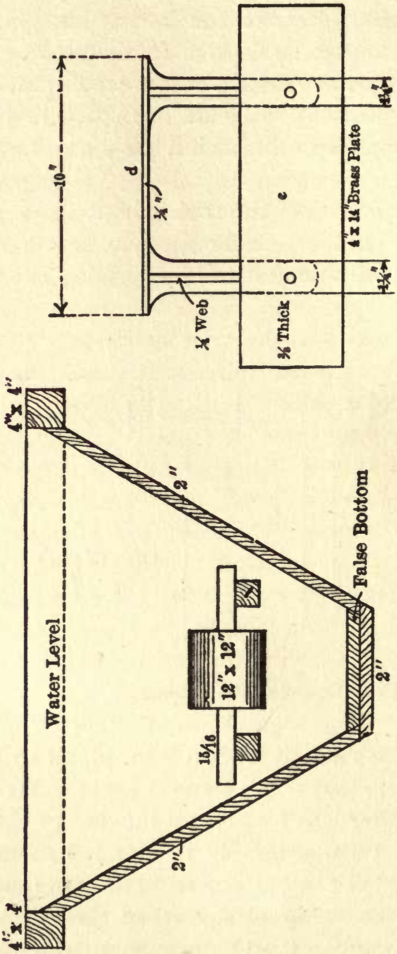



Fig. 40. — Section of Sand-box.

the tank is filled with water; also, any uneven settling can be readily remedied.

In the center of the bottom of the tank is cut a 6-in. hole, to which, on the under side, is bolted a 6-in. tee, to one leg of which is attached a gate-valve that can be opened in emergencies, and

to the other, by reducers, a 2-in. hose, which is carried out past the chime and raised some 8 ft., where the end is closed with a wooden plug in the center of which is a 0.5-in. nipple.

Supported vertically in the center of the tank is a long tube or box open at both ends, 12 by 12 in. in section, extending from above the water level down to within 2 ft. of the bottom, and directly over the 6-in. slimes outlet. Into the upper end of this tube the slime and water from the sand tank is introduced; it is delivered at the bottom at the apex of the cone, whence it rises evenly without irregular currents. With the velocity decreasing as the square of the ascension, the slimes are gradually dropped and are drawn continuously from the 0.5-in. nipple.

The velocity of discharge, and consequently the quantity of pulp drawn from the bottom of the tank, can be accurately gaged to suit the needs by raising or lowering the discharge end of the hose, and thus decreasing, or increasing, the head under which it works. This is an important detail, as no valve or jig-gate can be set to a small discharge of slimes, without sooner or later causing a disastrous stoppage that blocks the whole system. In our instance we found an 8-ft. head gave us a desired discharge of thick slimes, of about the consistency of molasses.

Under this head, and with a 0.5-in. orifice, clear water would have been discharged at a rate of 12,367 gallons per 24 hours, using the coefficient of discharge of 0.61. In our case the friction of the thickened slimes, through the 2-in. hose and small orifice, reduced the discharge to 6500 gallons per day of slimes, with two parts of water to one of solid matter.

These slimes are drier than those recorded by Mr. Denny in his practice, as 4 to 1 from the first operation, and I think this is mainly due to the use of the hose discharge. We anticipated discharging our slimes with the proportion of 2 to 1, but as there were more slimes than anticipated, it led to the discovery that when reduced to this thickness they settled, and gave a covering of clear water in time remarkably short in view of their former obstinacy. Accordingly we were led to re-treat them, which should have been done by the same system, but being driven to act quickly, we made a 6- by 8- by 6-ft. V-box, which, taking so small a stream of pulp, discharged clear water, and settled slimes in the bottom as a thick mud, from where it was drawn intermittently by an attendant who took this as one of his miscellaneous duties.

Ten tons of slime tailings were thus discharged here, that would average about one part water to one of solid matter, which caused a loss of 10 tons of water, or 2400 gallons per day. In summation there are losses as follows:

| | Gallons |
|---|---|
| 5 tons of lead concentrates | 180 |
| 15 tons of zinc concentrates | 900 |
| 50 tons of coarse tailings | 3000 |
| 10 tons of slime tailings | 2400 |
| Total | 6480 |

In such a climate the evaporation may be considered, but it is not as much as is generally supposed, and, in our case, we estimated as nearly as possible that we were losing 6500 gallons per day, distributed as above. With 216,000 gallons in circulation, this loss of 6500 gallons represents a loss of only 3 per cent.

Various attempts have been made to claim a saving equal to this, by the use of huge and cumbersome V-boxes, as, notably, at Cananea, but, from casual observation, it could hardly be credited.

I have noticed that, where apparently muddy water was flowing over a wide weir, there was generally a thin film of clear water on the surface. Whether this is due to any property of the surface of water peculiar to the surface only, as in the capillary attraction shown on the sides of a vessel of water, or on the bead over the gager's hook, I do not know, but to draw clear water from a light slime it is necessary to reduce the velocity of discharge by extending the cross-section of discharge to its limit of breadth, so that the depth of the cross-section of discharge reduces the water to a practical film; which brings us to the same thing.

In the case of our 16-ft. tank, the loss of 6500 gallons left 209,500 gallons in circulation, which was cleared in 24 hours. As our discharge was over the entire circumference of the upper rim of the tank, this was equivalent to discharging over a 50-ft. weir. Such a quantity of water discharging, in this time, over a 50-ft. weir would flow with a depth or head of only 0.1875 in. This was the depth of our film, as near as it was practical to measure. Simple hydraulic formulas will show that from a head of 0.1875 in., under the above conditions, we will have a velocity of approach of 0.42 ft. per second, or 1500 ft. per hour.

These figures I consider important for a justifiable anticipa-

tion of good work. I know that the tank would do nothing like the same work if the discharge were drawn from a 12-in. opening at any one point of the rim. If this should be done, we would have a 12-in. weir, which, to discharge the above amount of water in the same time, would require a depth or head of 2.4 in., which would draw from below what I would call the safe film, and would also create a velocity of 1.6 ft. per second, or 5760 ft. per hour.

In this latter case, the depth over the weir would be 13 times as great, and the velocity of approach would be four times as great, which would again seem to accentuate the fact that the depth of discharge from the tank is of more importance than the velocity.

## REMOVAL OF SAND FROM WASTE WATER

By J. E. Johnson, Jr.

(December 31, 1903)

I have read with the greatest interest the article of Messrs. Butters and Crank in the *Engineering and Mining Journal* of December 5, and that of Jesse Scobey in the issue of December 10, both dealing with the matter of handling sand, especially its removal from streams of running water. The reason for this interest is that I had occasion to deal with this problem about seven years ago, although under slightly different conditions. The problem and its solution were both presented in an article in the *Transactions* of the American Institute of Mining Engineers entitled "An Apparatus for the Removal of Sand from the Waste Water of Ore-Washers," to which all who are interested in this subject are respectfully referred. It may not be amiss, however, as nearly six years have elapsed since the publication of that paper, to go briefly over the ground there covered, and outline the solution of the problem reached.

The problem was one of the removal of sand and fine iron ore, all of which passed through a 14-mesh screen from the waste water of an iron-ore washer. The material varied from the size of the mesh down to the finest silt and clay, held in suspension in the water, but a great deal of it was an iron-ore sand of about the size of ordinary beach sand, such as probably most of your readers familiar with mill practice will recognize as passing through a 14-mesh screen. This material was so heavy that its removal become almost a necessity, since it clogged the long troughs used to carry the water to the settling ponds, and caused them to fill up so that they overflowed unless the inclination was excessive, in which case the hight lost was a serious consideration. Moreover, it was recognized that the greater part of this material was iron ore, and it was hoped that a method of treating it so as to produce a material of commercial value would eventually be found, and this has since actually been done. Moreover, the bulk of this material in the settling ponds was most objectionable,

filling them up with great rapidity. The quantity of material of this nature handled is about 50 tons in 10 hours, and the water from which it is removed amounts to about 750 gallons per minute. None of the ordinary solutions of the problem which had then been made seemed to suit the conditions without promising endless wear, and, in fact, in adopting the traveling-belt plan of operations, Mr. Scobey states plainly that wear at certain points is to be provided for, and that it is much better to admit this and make preparations accordingly in the first place. The general plan of the sand-shoveler devised by me assumed as its fundamental principle that its bearings should be kept away from the sand and water. It was obvious that a vertical wheel would carry sand over and drop it down on its own bearings. It was equally obvious that a horizontal wheel would not remove the sand from the water at all. It was not obvious by any means, but was eventually found to be the case, that a wheel with an inclined axis was capable of elevating the sand and of being supported so that no sand or water whatever reached its bearings. The central idea of reducing the velocity of the water, so that it would drop the material it carries in virtue of its velocity, is not unlike that of Mr. Scobey's, but instead of the dragging action used by him, a true shoveling action is employed. Almost the only defect of the machine is the great difficulty of representing its action by a drawing, or even by a photograph. The axes of the wheels, of which there are two, stand at 45 deg. each side of the vertical line, and the general plane of the shovel blades stands at approximately 45 deg. with the axes of the wheels, so that in their lowest position they are approximately horizontal, and in their uppermost position they are approximately vertical. The consequence is that they go into the sand at a slight angle from the horizontal, pick it up from just above the bottom of the settling box, or tank, in which they work, move it diagonally through the water, at the same time raising in front and tipping down behind, an action which has become pronounced when the blades emerge from the water, so that the water picked up by them runs off the back edge while the sand stays on the front edge. The diagonal motion carries them gradually clear of the sides of the troughs and the angle with the horizontal is constantly increasing, so that when the blade is in its highest position it is vertical, and the sand has nothing to do but slide off clear of the

edge of the trough. The action of the machine both in delivering more of the sand, and delivering it in a drier condition, is greatly enhanced by making the shovels of perforated material. It is, of course, impracticable to get any material with holes fine enough to retain the sand and yet have the material stiff enough to support the stresses upon it. Accordingly the shovels are made of double thickness, the bottom one of $\frac{1}{8}$-in. steel thickly perforated with $\frac{3}{8}$-in. holes, the upper one of about No. 16 zinc perforated with $\frac{1}{16}$ in. holes. The fact is familiar to all that sand will flow as long as it has water in it, but sets very firmly when it becomes dry, and this construction of the shovels takes advantage of this fact, since the water drains out of the sand almost instantly owing to the large surface exposed for drainage, with the result that the sand sets quite solidly, like wet beach sand, and does not display any tendency to run over the back edge of the shovel, as it sometimes does with solid blades. The drive of the machine is through a horizontal shaft carrying a worm into which worm-gears on the upper extremity of each wheel-shaft engage, so that there are but three moving parts in the whole machine. As to durability, the machine requires a new set of zinc covers on the blades about every year and a half, and the worm which drives it was renewed about a week ago for the first time in about six years. The zinc sheets for the shovel blades cost something less than 50c. each, and the worm is an ordinary cast-iron worm worth not over $5 at the outside. These are all of the repairs which have been put on the machine in its 7½ years of constant operation, without any attention whatever except an occasional oiling.

It will be seen that the machine has to remove its sand from a very much larger quantity of water proportionately than the apparatus of Mr. Scobey, as 750 gallons of water per minute in 24 hours amount to 1,080,000 gallons, whereas Mr. Scobey had only about 214,000 gallons, and the sand removed by this machine is about 50 tons in 10 hours or 120 tons in 24 hours, where his handled 50 tons in 24 hours; or about 900 gallons of water per ton of sand in our practice and about 4280 gallons per ton of sand in Mr. Scobey's case. As the energy of the water varies as the square of its velocity, it is obvious that the recovery of the sand from the water would have been very much easier in his case than in ours. The capacity of the machine, is, to be sure, somewhat greater than needed by Mr. Scobey, but even if this size of

machine were used, its size would be an advantage, in that the velocity of water through the settling tank would be so much the less that the extraction of sand would be the more complete. I feel sure that this would be the case. In regard to cost of apparatus, Mr. Scobey gives no figures, but presumably there would not be a great deal of difference either way. When it comes to repairs and ease of maintenance, however, I am convinced that the sand-shoveler would come out far in the lead.

In regard to the Virginia City plant described by Messrs. Butters and Crank, it seems a pity that resort to so elaborate an apparatus should have been had to accomplish the end desired. As far as the distributor is concerned, there is nothing to be said, since that is not a very complex machine and should, from its description, be a very efficient one; but the method of recovering the sand from the water, while it may work, is enormously expensive in first cost, and more or less so in maintenance and in power requirements. It will be seen that the amount of sand treated in 24 hours is just about the capacity of the sand-shoveler as given above, and had the sand-shoveler been installed in an enlargement of the flume bringing the tailings from the waste dumps to the mill, the sand could all have been removed from the flume at one point without the necessity of settling tanks at all, and it could have been carried by a much simpler system of conveyors directly to the distributor. The attendance required would have been less, the work would, as far as can be determined from the description of the writers, have been done absolutely as well, and the saving in the first cost would have been enormous.

Considering the great number of cyanide plants, and other plants which have to handle tailings, now in existence, and the growing strictness of the powers that be in regard to contamination of streams, this problem of handling material of this class and removing it from running water becomes one of considerable importance. No less an authority than Dr. E. D. Peters said, during the discussion of my paper before mentioned, that this machine should have a field of usefulness in cyanide and similar plants, and while practically no effort has been made to introduce it into this field, the growing interest in the subject and the growing demand for something of this kind seemed to me a sufficient justification for calling the attention of the milling fraternity at large to this very simple and inexpensive machine.

## DISPOSITION OF TAILINGS

(December 17, 1903)

*Question.* — At a pyrites mine, situated in a populous district, it is proposed to erect a dressing-plant of about 100,000 tons yearly capacity. If no special precautions are taken, most of the finer tailings will pass into a small river, 5 to 6 km. long, and thence into a larger river on which is situated a valuable salmon fishery, and which is bordered by cultivated ground usually overflowed by the river when in flood. To prevent heavy damages, it is, therefore, necessary that the water from the small river be kept free from tailings. It is proposed to make, near the dressing-plant, a series of large settling-boxes provided with filters, having openings of 5 to 10 mm., in order to catch the pyrites; and also to make one or more settling basins in the small river. It has also been proposed to use filter-presses. As I am not sure that these measures would be sufficient, I should be obliged for some advice on this subject. The quantity of water, with the tailings from the dressing-plant, will be from 15 to 20 gallons per minute. The flow in the small river is 150 to 200 gallons per second. — H. H.

*Answer.* — This is a matter on which you should take special professional advice and make some experiments. If you do that, you ought to be able to determine what danger there may be and how it may be eliminated. We can only give you some general ideas.

The question of stream pollution frequently comes up. In the *Engineering and Mining Journal* of Dec. 5, 1903, page 864, our San Francisco correspondent referred to the fight now going on in Calaveras county, Cal., between the miners and the farmers, the latter complaining that the tailings from the copper ore dressing works at Campo Seco are polluting the water of the Mokelumne River, destroying the fish, and injuring vegetation along the banks. The State Fish Commission investigated the matter and found that nothing harmful was passing into the

river from the copper mines, and that although the water was discolored by material from the Gwin gold mine, where much blue, slaty rock is crushed, no fish had been killed, nor had fields which had been irrigated with the water suffered any damage at all. Samples of the water tested chemically disclosed the presence of no free acid.

The muddy water discharged from dressing works can be clarified perfectly; it is not necessary to resort to filter-presses, which would be, in all probability, far too expensive. It is not necessary even to let the clarified water go at all. The prime essential is to provide settling reservoirs of sufficient capacity.

At the mill of the Bullion-Beck & Champion Mining Company, at Eureka, Utah, crushing 200 tons, per 24 hours, of ore containing lead and copper minerals, the entire drainage was collected in a reservoir of 40,000 sq. ft. area, the sides of which were built of coarse tailings and made water-tight by the finest slimes. The clear water was drawn off from this reservoir into a tank 36 by 20 ft. in area, 7 ft. deep, whence it was pumped at the rate of 100 to 125 gallons per minute into a receiving tank, where it arrived almost free from sediment. No water was lost except by evaporation. At the washery of the Longdale Iron Company, in Virginia, the coarse sand is removed from the mill discharge by means of a mechanical device, and the slimy water is then run into reservoirs, through the banks of which it is allowed to filter; it escapes perfectly clear. This is done to avoid pollution of a stream.

Assuming that some such system be adopted in your case, a factor to be considered is the presence of acids or soluble salts in the water from your works, which may arise from the use of mine water or by fresh water taking up soluble matter from the ore; and, if present, whether they would be in sufficient quantity to injure the fish and vegetation; and if so, might they not be removed by some simple system of neutralization or precipitation. It would appear advisable to clarify your mill water before you allowed it to run into the small river at all, because then you would have a very much smaller quantity to deal with.

# PART VII

# MISCELLANEOUS

## RUBBER AND RUBBER BELTING

(March 9, 1901, and February 3, 1906)

Data as to the weight of rubber belting are seldom to be found in the catalogues of belt manufacturers, chiefly because it is so variable, depending upon the different conditions of manufacture. Belting consists of two parts, the duck and the rubber. The former is used in different weights in making belts of the same ply, and the weight of the rubber part is not only affected by the percentage and character of the adulterants that are used, but the weight of pure rubber is variable. Speaking generally, a good 8-in. four-ply belt ought to weigh about 1 lb. per linear foot. Probably all the rubber that is used is adulterated to some extent, now more than ever in view of the high prices which have been established by the greatly increased demand for rubber and the diminishing supply. This has led, among other things, to an active business in the recovery of rubber from old material, gum shoes and the like. Even the South American and African natives who supply the crude material have their tricks in the trade, one of which used to be to incorporate a stone or two in each cake of the crude; but the buyers circumvented that by cutting open the cakes upon purchase.

Para rubber is the standard. Rubber dealers have learned, however, to mix other rubbers in such ways and proportions as to produce an article which closely resembles the Para and would probably deceive any one but an expert. The adulterants commonly used in the manufacture of rubber goods are chalk, gypsum, calcined magnesia, asphaltum, barytes, litharge, talc, lampblack, and zinc white. Manufacturers are naturally reticent as to which of these they use and in what proportions. In a recent law case in England it was elicited from one of the witnesses that the material of which carriage tires were made was composed of 48.5 per cent. pure rubber, 5.4 per cent. sulphur, 6.7 per cent. barytes, 7.6 per cent. litharge, 20.7 per cent. chalk, 9.1 per cent. steatite (talc), and 2 per cent. lampblack. From this analysis

commercial rubber appears to be more of a mineral substance than it is a vegetable. The sulphur shown by the analysis is of course residual from the process of vulcanizing and is not properly to be considered as an adulterant.

Zinc white is largely used as an adulterant for rubber. The recent large increase in the consumption of that substance in the United States is said to be due to the rubber manufacturers rather than to the paint trade. It is claimed by some manufacturers that the quality of rubber is improved for certain purposes by the introduction of some of the above-mentioned mineral substances, its durability being in some way increased, and there appears to be more or less truth in their claims. Elastic bands are one of the purest forms in which rubber is used and every one knows how rapidly they lose their strength and break. Incidentally it may be remarked that an adulterant has recently been found that can be used to an important percentage in rubber for elastic bands without apparently affecting their elasticity.

Some specifications for belting call for a tenor in pure rubber of 45 to 55 per cent. However, the tensile strength of a good rubber belt is very much in excess of any pull to which it is likely to be subjected, though in this connection it should be noted that the weakest part is at the lacing, where the material must be strong enough to prevent the lacing from tearing through the relatively short distance between the holes and the ends. Although the mineral adulterants may not injure and may really improve the rubber employed in the manufacture of belting, their effect on rubber to be used for chemical and perhaps other purposes may be deleterious and should be well investigated. This may be done by incineration of a sample and chemical analysis of the ash.

The average weight of a good four-ply rubber belt, 7 in. wide, is about 16 oz. per linear foot. A very light 7-in. four-ply belt may weigh as little as 12 oz. per linear foot; a very heavy belt of the same dimensions may weigh as much as 20 oz. The weight of rubber belting varies to some extent with the quality of the raw material and the time of the year when manufactured, and it is consequently impossible to give exact figures. Some inferior rubber belts weigh more than first-class products because of the use of adulterated rubber, the adulterations employed being as a general thing heavier than the rubber itself.

The nature of these adulterants is a secret of the trade. It is well known, however, that zinc oxide, barytes, litharge, chalk, gypsum, talc, magnesia, asphaltum, and lampblack are used for this purpose. Experts in the manufacture of goods are able to identify adulterated rubber by its appearance and physical characteristics. The extent to which adulteration has been practised may be determined from the comparative weight of the ash of various samples after incineration.

The best rubber belting is made with 32-oz. duck. A 7-in. four-ply belt made of 32-oz. duck weighs about 20 oz. per linear foot; a 7-in. four-ply belt weighing 16 oz. per linear foot is made from 28 to 30-oz. duck; a belt of the same dimensions weighing only 12 oz. per linear foot would have only 26-oz. duck.

## COAL-DUST FIRING

BY C. O. BARTLETT

(December 31, 1903)

Three conditions are requisite for the successful combustion of coal in dust form, viz.: uniformity of moisture, uniformity in size of grain, and proper regulation of the supply of air. Uniformity in moisture is best attained by drying the coal. As to size, it is best reduced so as to pass an 80-mesh screen. Control of the air supply is best secured by introducing it by means of a fan or blower. If these three conditions are met, perfect combustion of the coal may be attained, with the results of no black smoke, no cinders and very little ashes, and a saving of practically 40 per cent. in the consumption of coal.

For the drying of the coal, a rotary cylinder, heated externally, is recommended. In some cases the products of combustion are allowed to pass first on the outside of the drier, and then through the inside. When this is done, it is safe to reckon an evaporation of 10 lb. of water per pound of coal burned. Otherwise, the ratio will be about 8: 1. Estimating on a basis of drying 40 tons of coal per day and an expulsion of 5 units of moisture, with coal at $2 per ton, the cost of drying will be less than 12c. per ton. With a good mill, requiring about 25 horse-power, 4 tons of dry coal can be pulverized per hour. The kinds of mills at present in use for this purpose are the French buhr mill, the Huntington, and the ball- and tube-mills. It is safe to reckon on a cost of 10c. per ton for pulverizing. The cost of elevators, conveyors, bins, running the blower or feeder, repairs and renewals, and interest on the investment, ought not to come to more than 10c. per ton, making the total cost only 32c. per ton of dry coal. The approximate cost of a plant for firing 4 tons of coal-dust per hour is as follows: Drier, $1500; pulverizer, $2500; feeders, $1000; conveyors, elevators, bins, etc., $2000; total, $7000. Besides the economy in coal, dust-firing presents the further advantage of a

saving in labor. Laborious handling of coal into the furnace is obviated; there is not more than one-third as much ashes to be handled, and there are no cinders at all. The economical application of coal-dust firing implies, however, the use of a fairly large quantity of coal. It would not pay to put in the necessary plant for a consumption of less than 1 ton per hour.

In coal-dust firing, the arrangement of the combustion chamber of the furnace is an important consideration. In one experiment the Rowe system was used, which consists in blowing the pulverized coal, by means of an ordinary fan, against an arch, burning the coal in suspension. The arch becomes very hot and immediately ignites the coal; when the proper quantity of air is used, combustion is perfect. It was found that a 2-oz. pressure was best for this purpose. In order to withstand the very high temperature, special brick should be used in the construction of the arch. There is a German invention for protecting the arch with a lining made of carborundum dust, which, it is said, will withstand the highest temperature until worn away mechanically.

## INTERNALLY FIRED BOILERS

(August 8, 1903)

Internally fired boilers of the Scotch type are finding increased use in the United States for general purposes of steam generation. Many of the newer metallurgical plants are equipped with this form of boiler. It is somewhat more expensive than the ordinary horizontal return tubular boiler, allowing for the brick setting of the latter, which the internally fired boiler does not require, but the additional cost is not so high as to offset the advantages in many cases.

Among the advantages of the internally fired boilers is safety against explosion. It is said that there is no case on record of the explosion of a boiler of this type. The boiler, not being enclosed in brick work, is subject to inspection from the outside at all times, and the plates of the shell are not exposed to the direct heat of the fire, wherefore there is but little deterioration in the plates of the outer shell or the joints thereof. The danger of bagging sheets, which is apt to occur in any externally fired boiler, due to an accumulation of scale or sediment on the fire-sheet, is also eliminated. The lower part of the internally fired boiler is not exposed to the fire at all, and any deposits collecting in the form of a slush or mud are readily blown out. It is claimed also that the tendency for scale to adhere to the furnace is reduced to an extent that will not cause any trouble, the expansion and contraction of the corrugated surface of the furnace having the effect of loosening the scale. With respect to the small fire-tubes, the conditions of the internally fired boiler are substantially the same as those of the ordinary horizontal return tubular type.

The chief advantages of the internally fired boiler are economy in fuel and lower cost of maintenance. A saving in the latter respect may be expected in almost all cases. There is no brick setting to maintain, and, as stated above, there is less likelihood of deterioration in some of the parts of the shell. The saving in fuel is determined by the fact that the fire is entirely surrounded

by water to be heated, wherefore no loss is suffered through radiation from the fireplace. With many kinds of fuel this may constitute a rather important advantage. This advantage may be offset, however, with poor grades of coal by the limited volume of the fireplace, which does not afford sufficient space for the gases to mix in order to insure complete combustion before the temperature is cooled below the ignition point.

The internally fired boiler is attractive because of its neat appearance in place and the simplicity of erecting it, the boiler being delivered in a complete condition and requiring nothing but a few small piers for its foundation. The floor space required is also somewhat less than for the ordinary return tubular type.

The advantages of the cylindrical boiler, with internal corrugated fire-box, and its superiority to the ordinary locomotive type boiler have been recognized by railroad men. The use of such a boiler for a locomotive was first suggested by Bela Ambrosovics on the Hungarian State railroads about fifteen years ago. The idea was taken up by Herr Lentz, on the Prussian State railroad, and Herr Scheffler, on the Saxon State railroads, about 1891, and at about the same time George S. Strong used it in an experimental locomotive built by the Rhode Island Locomotive Works. Mr. Strong's idea was not a success, but its failure was due to other causes than the fire-box. The plan was taken up again by Cornelius Vanderbilt, Jr., on the New York Central, some two years ago, and since that time boilers of this type, with corrugated fire-boxes, have been built for a number of roads, and have proved highly successful. We may add that this type of boiler is especially adapted for the use of liquid fuel, which is becoming quite common in California and in the Southwest.

## ACCIDENTS TO MOTORS AND DYNAMOS [1]

By A. C. Cormack

(October 3, 1903)

An analysis of the breakdowns occurring during the last four years to several thousand motors and dynamos, ranging in size from 0.5 horse-power to 800 kilowatts, has been made. The machines taken into consideration had been working under conditions better than the average, and consequently the results obtained must be regarded as considerably above the average. Motors driving coal-cutting machines have not been included.

The relative frequency with which various portions of dynamos and motors were damaged in breakdowns was as follows: *Mechanical portions :* broken shafts, 2.35 per cent.; binders and fasteners, 9.40 per cent.; bearings, 8.60 per cent.; other mechanical parts, 4.70 per cent.; total, 25.05 per cent. *Brush-gear, connections, terminals, etc. :* Failure of brush insulation, 2.35 per cent.; connections and terminals, 3.10 per cent.; mechanical portion of brush gear, 1.56 per cent.; total, 7.01 per cent. *Field magnet coils :* Coils burnt out, 5.50 per cent.; coils earthed, 3.10 per cent.; coils gone altogether, 1.56 per cent.; total, 10.16 per cent. *Armatures :* Coils short-circuited and burnt out, 29 per cent.; coils earthed (that is, failure of insulation of frame), 19.50 per cent.; breakage of wire, 15.60 per cent.; joint failures, 8.60 per cent.; binders burnt, 3.10 per cent.; driving horns, 2.35 per cent.; total, 33.60 per cent. *Commutators :* Failure of insulation, 10.15 per cent.; mechanical failures, 10.95 per cent.; surface fusion, 12.50 per cent.; total, 33.60 per cent.

When several parts failed in a single breakdown, each is included in the above summary, except those cases where the failure of one part was certain to follow the failure of another. Of the accidents in which the field magnet coils were damaged,

[1] Abstract of a paper on "Electric Plant Failures, Their Origin and Prevention," read before the British Institution of Civil Engineers, 1903.

the larger number were cases in which the insulation between the coil and core gave way. The failures due to the entire burning out of the coils, i.e., arcing from series to shunt on the same coil, or arcing between two separate coils, occurred most frequently in bipolar machines, where the coils often touch. The failure of armatures is the most serious danger, usually taking the form of a burning out of coils. Coils earthed were generally due to destruction of the insulation. Commutators are frequently damaged, failure of insulation being the most frequent trouble. The mechanical failure of commutators is due to bursting, fracture of commutator-lugs and failure of keying.

The points of origin of breakdown, together with the percentages of the frequency with which the various parts failed, were as follows: *Mechanical portions:* Shafts, 0.78 per cent.; bearings, 8.60 per cent.; general, 1.65 per cent.; total, 11.03 per cent. *Brush-gear, leads, and terminals:* Mechanical, 3.90 per cent.; insulation, 2.35 per cent.; leads and terminals, 2.35 per cent.; total, 8.60 per cent. *Field magnets:* failure of coil insulation, 7.80 per cent.; breakage of wire, 0.78 per cent.; total, 8.58 per cent. *Armatures:* Core and slot insulation, 12.50 per cent.; end insulation (that is, separating bridges), 5.50 per cent.; coil insulation, 11.70 per cent.; joints, 7 per cent.; fasteners and binders, 1.65 per cent.; binder insulation, 7.80 per cent.; driving horns, 1.65 per cent.; total, 47.80 per cent. *Commutators:* Insulation, 9.40 per cent.; fastening and keying, 5.50 per cent.; total, 14.90 per cent. *Starters,* 4.70 per cent.; *unknown,* 3.13 per cent.; *various,* 1.56 per cent.

Shaft breakdowns were caused generally by the armatures coming out of center through wear of the bearings. Bearings frequently wear upward, this sometimes occurring in cases where the armature has been placed nearer the top of the race than the bottom in order that the upward pull of the magnets may relieve the pressure on the bearings. Owing to the liability of this wear to escape attention, it is somewhat dangerous. Commutators were responsible for 14.9 per cent. of the breakdowns. Commutator breakdowns are of two classes. The first, in which the insulation has failed, occurs most frequently between the bars and supporting rings, and results often in the burning out of the armature windings. Defective fastening and keying of the commutator constitute the second class of accidents to which

this part of the machine is subject. When the keying is defective the commutator is driven by the armature wires, and sometimes the commutator and the armature have a slight relative motion on the shaft. This causes breakages of the wires joining the armature to the commutator.

The conclusions which have been reached as to the causes of breakdowns are summarized as follows: *Constructional:* Bad design, 18.36 per cent.; perishing of insulation, 7.40 per cent.; bad workmanship, 13.60 per cent.; total, 39.36 per cent. *Conditional:* Overloading, 1.37 per cent.; over-rating 1.56 per cent.; unsuitable, 0.78 per cent.; total, 3.71 per cent. *Maintenance:* Dust and damp, 7.40 per cent.; rough usage, 1.56 per cent. Defective attention other than above, 22.50 per cent.; total, 31.46 per cent.; *accidental and unavoidable,* with reasonable care in construction and working, 7.05 per cent.; *unknown,* 13.10 per cent.; *caused by faults in accessories,* 5.48 per cent.

Comparatively few of the faults were found in the mechanical portions of the machine. These usually took the form of insufficient keying, bad arrangement of bearings, and other ordinary mechanical defects. There is a tendency to design machines having higher rises of temperature than experience has shown to be compatible with reasonable durability of the machine. Defects from dust occur most frequently in semi-closed motors, for which the provision for cleaning is usually insufficient.

## ALLOYS FOR BEARING PURPOSES[1]

By G. H. Clamer

(September 12, 1903)

A good alloy for bearing purposes must consist of at least two constituents, namely, a hard one to support the load and a soft one to act as a plastic support for the harder grains. If the bearing were always in perfect adjustment with respect to the journal a hard, unyielding alloy would be the best, since the harder the alloy the lower is the coefficient of friction, generally speaking. It is found, however, that, owing to the imperfect nature of the surface in practice, a hard, unyielding alloy which cannot mold itself to the irregularities of the journal will cause a concentration of pressure upon a few high spots, with the result of rapid heating and abrasion. Soft metal bearings, on the other hand, are apt to give rise to unduly rapid wear of the journals; but whether this be due to the imbedding of grit in the bearing surfaces, or to the fact that the metal itself has a dragging nature, is uncertain. The excessive collar wear of journals is, however, undoubtedly caused by the lead lining lapping out of the fillets.

The cheapest of the white metal alloys is lead and antimony. These metals alloy in all proportions, but the eutectic mixture is composed of approximately 87 per cent. lead and 13 per cent. antimony. This has been adopted by the Pennsylvania Railroad. As the percentage of antimony increases the alloys become more brittle; with more than 25 per cent. antimony they are unsafe to use. Charpy considers that the alloys containing between 15 and 25 per cent. antimony are the best constituted for bearings, the free antimony forming the necessary hard constituent imbedded in the plastic eutectic. However, it may be pointed out that the alloys containing free lead also possess the necessary structure of a hard constituent, in this case the eutectic, imbedded

[1] Abstract of a paper in the *Journal* of the Franklin Institute, 1903, CLVI, pp. 49-77.

in the more plastic lead; although the friction of such alloys is higher than that of those containing an excess of antimony, the wear is much less. Lead is the best wear-resisting metal known; with additional antimony, increasing hardness and brittleness, the wear augments, owing to the breaking off of the harder particles.

Tin added to lead and antimony imparts rigidity and hardness without increasing the brittleness. It is desirable, therefore, when high pressures have to be carried. Babbitt metal, which is regarded as the standard of excellence, consists of 89.1 per cent. of tin, 7.4 per cent. of antimony and 3.7 per cent. of copper. It is the most expensive of the white metals, and in the majority of cases may be replaced by a cheaper alloy.

With respect to the bronzes, composed of copper, tin, and lead, it is found that the rate of wear diminishes with decrease of tin and with increase of lead. Dudley's "Ex. B." alloy, consisting of 78 per cent. copper, 7 per cent. tin, and 15 per cent. lead, was formerly believed to represent the minimum of tin and the maximum of lead possible in practice without danger of liquation of the lead occurring in the mold. Experiments have since shown, however, that while a certain proportion of tin is necessary to prevent liquation of the lead, and also to give the alloy sufficient compressive strength, a larger proportion is very detrimental. Alloys containing 5 per cent. tin and 30 per cent. lead have been produced without difficulty, but if the tin exceeded 6.10 per cent., castings containing 30 per cent. lead could not be obtained, and if the tin exceeded 7 per cent., not more than 20 per cent. of lead could be introduced under practical conditions. An alloy finally invented was composed of 64 per cent. copper, 5 per cent. tin, 30 per cent. lead, and 1 per cent. nickel. This is known as "plastic bronze." Nearly 4,000,000 lb. of it have been successfully made during the last three years, in castings weighing from 1 lb. to over 1000 lb. The castings are sharp and clean and can be readily machined.

## COST OF SMALL POWER PLANTS

(September 5, 1903)

Some recent bids on high speed engines of 100 to 300 i. h. p., by high-class manufacturers, have ranged from 9c. to 12c. per lb., f. o. b. factories, the former price being on the larger engines and the latter on the smaller. These prices figure out to $11@$18 per i. h. p. An engine of 150 horse-power costs about $12.50. A good Corliss engine of the same power is obtainable for about $12 per i. h. p., or 7.5c. per lb. Good horizontal, return tubular boilers of about 100 horse-power, designed for steam pressure of 100 lb., can be had for $7@$8 per horse-power. Internally fired boilers of the same capacity cost about $12.50 per horse-power, but the difference is partially offset by the less cost of setting. The water-tube boilers are the most expensive in first cost. A good feed-water heater can be purchased for about $1 per boiler horse-power. The total cost of a steam power plant of 300 horse-power capacity will be about $50 per horse-power.

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The ordinary pile-driver cannot be very much improved in the matter of capacity. It is true that steam drivers have been, and are being, successfully used; but the shorter period taken in driving does not materially affect the total capacity of a crew. The main cause of the limited output in driving piles is the length of time necessary to adjust the pile and the machine to the position required; this is irrespective of the type of driver used, and the rapidity of action of the latter has so slight an effect upon the total time spent that the increase in the number of piles driven per day is very small.

## CONSTRUCTION OF WOODEN WATER TANKS

(December 5, 1903)

Water tanks for fire protective purposes, which often do not receive ordinary care and supervision, frequently collapse, sometimes with considerable damage. The fire underwriters have been led, therefore, to study the causes of such collapses and the remedy. Their conclusions are of value in specifying the proper construction for wooden tanks for any purpose.

The most frequent cause of collapse is corrosion of the hoops. The ordinary hoop is a flat wrought-iron band, $\frac{1}{8}$ to $\frac{1}{4}$ in. thick, and of varying widths. They are seldom painted, either before or after putting on, and are subject to corrosion both from the outside and the inside. Outside corrosion is easily observed, but not so the inside corrosion; and the real condition of the hoops is seldom known until collapse occurs. Inside corrosion is due chiefly to moisture, which works into and through the staves of the tanks. Many hoops corrode sufficiently to fall off by their own weight. The only way to determine the condition of flat hoops is to reinforce the tank with additional hoops, and then remove the old ones for examination.

Flat hoops, when properly cared for, will give satisfactory service, but round hoops are much to be preferred. With round hoops, corrosion from the inside acts only on a small surface, as compared to the flat hoop, and ordinary inspection will detect the corrosion before it has progressed sufficiently to weaken the hoop. Round hoops have the further advantage that swelling of the tank is less likely to burst the hoops, since a round hoop will indent itself into the wood.

Hoops should be made of wrought iron of good quality, with tensile strength of not less than 50,000 lb. per sq. in. In determining the size, a factor of safety of 5 or 6 should be used, and allowance should be made for the swelling of the staves, which will strain the hoops more than the actual pressure of the water in the tank. Hoops should never be less than $\frac{3}{4}$ in. in diameter. They should be made without welds, and should be thoroughly painted before and after erection.

## PIPE LINE CONSTRUCTION

(October 10, 1903)

There are said to be approximately 25,000 miles of pipe line laid for the conveyance of natural gas in the United States. These are of various sizes, ranging from 2 in. in diameter up to 36 in. For the transportation of large quantities of gas, or even comparatively small quantities when the line is a long one and pumping is not resorted to, the pipes have to be of considerable diameter. Thus the transportation of 4,000,000 cu. ft. of gas per 24 hours (166,667 ft. per hour) a distance of 20 miles, with intake pressure of 200 lb. and discharge of 20 lb., requires a 6-in. pipe. Reckoning 25,000 cu. ft. of gas as equivalent to 2000 lb. of coal, 4,000,000 cu. ft. of gas per 24 hours corresponds to only 160 tons of coal. The construction of a 6-in. line will probably cost about \$4750 per mile, or \$95,000 for 20 miles. Allowing only 10 per cent. for interest and depreciation, the daily charge is \$9500 ÷ 365 = \$26, which would be equivalent to paying for the carriage of coal at 0.8c. per ton-mile. When the intake pressure, i.e., the pressure at the gas wells, falls below 200 lb., either a pumping plant must be put in or a larger line must be put down. Remoteness from sources of the gas therefore increases the cost of the latter very rapidly. The construction of a pipe line of large diameter is an expensive undertaking, the larger sizes exceeding in cost that of a first-class railway line.

Pipe lines for the transportation of natural gas are commonly constructed of wrought-iron pipe, technically known as "line pipe," which is tested up to 1500 lb. Line pipe is made with longer couplings than ordinary pipe. The standard sizes range from 2 in. to 12 in. diameter. The smaller pipe lines, say 10 in. in diameter or less, are usually laid with the standard, screw-joint line pipe, but lately plain-end pipe with couplings and rubber packing has become very popular, and is to a large extent taking the place of screw-joint pipe, it being equally cheap in first cost and more readily laid. The pipe line should be buried under

ground, so that the distance from the surface of the ground to the top of the pipe will be 18 to 20 in. This puts the line out of the way and makes it comparatively free from the expansive and contractive force to which it would be subject if it were exposed on the surface of the ground to the direct heat of the sun. The ditching is not very expensive; in ordinary soil it ought not to cost much more than 6c. per lineal foot. The pipe should be well protected with coal tar on the outside before laying. Expansion joints should be put in from time to time, say at one-half mile intervals, and also a certain number of gate-valves and tees to permit connections to be made at points where they may be required. A well-laid pipe line suffers comparatively little depreciation in value, and the pipe, after having been in the ground for four or five years, may be taken up and sold at a discount of only 15 to 25 per cent. from the first cost, according to the demand at the time. Second-hand pipe in large quantity is always more or less in request.

Sizes of pipe from 10 in. to 2 ft. in diameter are frequently laid with the Converse joint, in connection with which the pipe has plain ends, the connections being made by heavy cast-iron sleeves, or hubs, fitted with molten lead, which is calked into the space between the pipe and the sleeve. This joint has the disadvantage that the packing may be loosened by the settling of the pipe and the movement due to changes in temperature. However, by the addition of a rubber packing pressed against the outside of the joint by means of iron clamps, a satisfactory connection can be made. Pipe lines of this construction have operated under a pressure of more than 300 lb. per square inch. For pipe lines of more than 2 ft. diameter, cast-iron water pipe or riveted steel pipe are put down. A line of riveted steel pipe, 36 in. diameter, extending 20 miles southward from Pittsburg, Pa., cost approximately $50,000 per mile. At present, Pittsburg is receiving gas from points in West Virginia more than 100 miles distant, while the lines which supply Chicago, Toledo, and Cleveland are considerably longer, the gas in some cases being conducted 200 miles.

## TRANSPORTATION OF GAS BY PIPE LINES

(February 4, 1904)

*Question.* — On page 541, issue of the *Engineering and Mining Journal* of October 10, I notice an article entitled "Pipe-line Construction," from which I quote as follows: "Thus the transportation of 4,000,000 cu. ft. of gas per 24 hours (166,667 ft. per hour) a distance of 20 miles with intake pressure of 200 lb. and discharge of 20 lb. requires a 6-in. pipe."

I am very much interested in the matter of transportation of gas through pipe lines under high pressure, and have made quite a study of what information I have been able to secure on this subject; from all of which I have worked up a formula which seems to me to be the best and most complete of anything that I have found, although there seems to be a great poverty of information on this subject. Working the problem in the above quotation by this formula, and on the assumption that the natural gas referred to has a specific gravity of 0.50, the results should widely differ from those stated by you, and I therefore beg to inquire if the information given is the result of observation from actual practice, or whether it is worked from a formula. If so, I will be pleased to learn the formula used. — L.

*Answer.* — The capacity of the pipe line under the conditions specified was computed by the formula

$$Q = 41 \sqrt{\frac{d^5(P^2 - p^2)}{l}}$$

in which $Q$ is the quantity of gas in cubic feet per hour, $d$ the diameter of the pipe in inches, $P$ and $p$ the absolute pressure of the gas in pounds at the inlet and outlet, respectively, and $l$ the length of the pipe in miles. The factor 41 is a constant. It is sometimes taken as 42 and sometimes as 40. It should really be a trifle more than 41, being based on the figure 3000 when $l$ is stated in feet; extracting the square root of 5280 and dividing 3000 by the quotient gives 41.1+. This formula is used by some engineers to arrive at the volume of gas of 0.65 specific

gravity. According to it, a 1-in. pipe, 20 miles long, with pressure of 200 — 20 lb. (above atmospheric) would deliver about 1959.4 cu. ft. per hour. A 6-in. line would deliver 95 times as much, or 186,143 cu. ft. This is more than would be worked out directly by the formula for a 6-in. line, the result of which would be 171,495 cu. ft. This is because the multipliers determined by experiment show that the capacity of a pipe increases actually in a little greater ratio than the square root of the fifth power of the diameter.

Another formula that is employed is

$$Q = 40d^2 \sqrt{\frac{d(P-\rho)}{Dl}}$$

in which $D$ is the specific gravity of the gas; save for that, and the use of the factor 40 instead of 41, it is the same as the first formula, but the introduction of the factor of specific gravity, which is a decimal, in the denominator under the square root sign gives a larger quotient and consequently higher results.

Both these formulas give very much higher results than those which are commonly given for the flow of coal gas in pipes, which, according to various authorities, are as follows:

| *King* | *Molesworth* | *Gill* |
|---|---|---|
| $Q = 1350\sqrt{\frac{d^5h}{sl}}$ | $Q = 1000\sqrt{\frac{d^5h}{sl}}$ | $Q = 1291\sqrt{\frac{d^5h}{s(l+d)}}$ |

In these $d$ is the diameter of the pipe in inches, $h$ the pressure in inches of water, $s$ the specific gravity of the pipe, and $l$ the length of the pipe in yards.

Gill's formula is said to be based on experimental data, and to make allowance for obstructions by tar, water, and other bodies tending to check the flow of gas. An experiment made in London with a 4-in. pipe, 6 miles long, with gas of 0.398 specific gravity and 3-in pressure, gave a result which corresponded closely with Molesworth's formula. (See Kent's "Mechanical Engineer's Pocket-book," p. 657.)

The figure 0.50 is rather low for the specific gravity of natural gas, which is commonly assumed at 0.60 to 0.65. The specific gravity of methane is 0.533, while that of ethane is 1.034. According to S. A. Ford, the specific gravity of the gas found in the vicinity of Pittsburg, Pa., was about 0.65, as a rule.

## MAKING PIPE JOINTS

(April 4, 1904)

At the meeting of the American Society of Heating and Ventilating Engineers, Jan. 19, 1904, there was a discussion of the question: "In making screwed joints, is it better to use some compound, or to make them up iron to iron with no compound?" In this connection, several bits of information as to why it is desirable to use some compound were given. Pipe fittings and the threads on pipe ends are not always uniform, even when coming from the same maker. They usually rust and accumulate dirt before being used, so that they go together hard; threads do not fit precisely, and fittings may leak, particularly under high pressure; also, threads not served with compound tend to cut when screwed together. Moreover, after the joint is made, the threads may rust together and make it difficult to unscrew the pipe. Red lead was formerly much used as a pipe compound, to lubricate, preserve, and make a tight joint, but nowadays most steam fitting is done without red lead. Graphite and oil, mixed to the consistency of table oil, are now frequently used, and are efficient both for lubricating and making a tight joint. The compound should always be put on the male screw, and not daubed into the female end, to prevent its being crowded to the rear of the joint, and thus reducing the clear opening of the pipe. It is necessary to keep a constant watch over workmen to prevent them from putting compound into the female end, or putting it on too thick; a thin coating is sufficient to get the desired effect. Black asphaltum, about as thick as melted butter, is also used for making pipe joints, with successful results.

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